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COAL DIVISION

1931

Wm. R. Chedsey.

CONTAINING PAPERS AND DISCUSSIONS PRESENTED BEFORE THE DIVISION AT PITTSBURGH, SEPT. 11–13, 1930; FAIRMONT, MARCH, 26–27, 1931 AND NEW YORK, FEB. 16–19, 1931

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CONTENTS

FOREWORD

This second Coal Division volume contains the papers on coal and allied subjects presented before the meetings held in the latter part of 1930 and the first half of 1931.

Five papers on subsidence and instantaneous outbursts of gas and one on ventilation were presented at the meetings of the technical committees dealing with those subjects, and are included here so that all papers relating to coal mining will be preserved together.

The paper on stream pollution brings the literature of this important subject up to date. The mining papers deal largely with operating problems of interest to the industry now, and the papers on cleaning and preparation are the results of work in these subjects which are daily assuming more importance.

The paper on gas and coal economics compares the relative values of the two fuels under varying conditions and presents some ideas of the combustion of the fuels which will not meet unanimous agreement.

At the Pittsburgh meeting in October, 1930, a Junior Branch was formed with the idea of interesting the younger members more in the work of the Institute, and it is hoped to establish more of these in the larger communities.

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Air Cooling to Prevent Falls of Roof Rock

By J. H. Fletcher* and S. M. Cassidy,* Chicago, Ill.

(New York Meeting, February, 1931)

Air has been cooled, heated, washed, humidified and dehumidified for many purposes and in many industries. At a number of metal mines air is conditioned to reduce the high humidity and unbearable heat found in deep workings, and at some mines air is washed to allay siliceous dust. Steam is introduced into the intake air at many coal mines, particularly in winter, for the purpose of keeping loose coal dust so moistened that the hazard of an explosion is lessened.

This paper will describe a further successful application of air conditioning in a coal mine, where, for over three years, all intake air has been cooled to a constant temperature for the sole purpose of preventing roof falls.

Effect of Warm Weather on Mine Roof

It is generally known by mining men that some, if not all, coal mines have more trouble from bad roof during the warm summer months than in winter. Also, it has been observed that roof on intake airways is usually worse than on the return entries, after the air has circulated through the mine. Some mines and some coal seams are much more affected than others.

In the Clinton district of Indiana, the No. IV coal seam has a good, firm, gray shale roof except during the summer months, when it deteriorates badly and falls in small slabs 1 to 6 in. thick. Timbering is not very efficacious during this period because in time the roof will shear around the supporting prop or crossbar and either dislodge the timber or leave it supporting only a few inches of crumbly shale (Fig. 1). This action is renewed each summer, choking airways with fallen slate, tearing down trolley wires, hampering haulage, endangering men and adding considerably to the deadwork cost. Only when mines have been worked for a period of years and have been advanced a long distance from the shaft do the active workings become comparatively free of this trouble, and even then the intake entries are still a source of much deadwork.

Such was the annoying situation at the mine of the Saxton Coal Mining Co. operated in the No. IV coal just north of Terre Haute, Ind. During the months of warm weather it was possible to stand at any quiet spot in the mine and within 10 min. hear at least two or three falls,

* Allen & Garcia Co., Consulting and Constructing Engineers.

usually not large but, of course, dangerous and troublesome. It was, indeed, hard to find a place in the mine where the roof would not sound drummy when tested.

Furthermore, during summer the roof, ribs, rail and trolley wire were always dripping wet, because the hot outside air was cooled in the mine until it became supersaturated and therefore deposited its excess moisture. In the winter, the opposite was true.

There are two generally held theories to explain the deterioration of mine roof in summer; one is that moisture acts on lime or some other ingredient of the roof and causes it to crack and fall, another is that

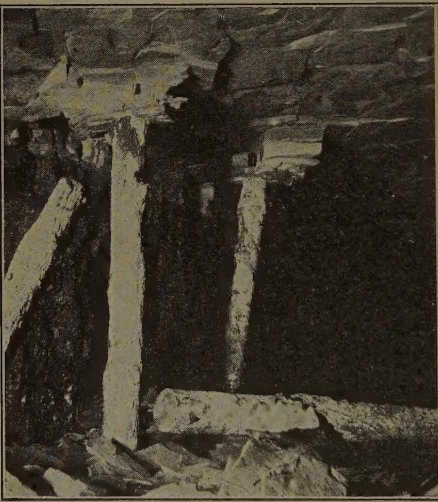

Fig. 1.—Roof affected by warm air.

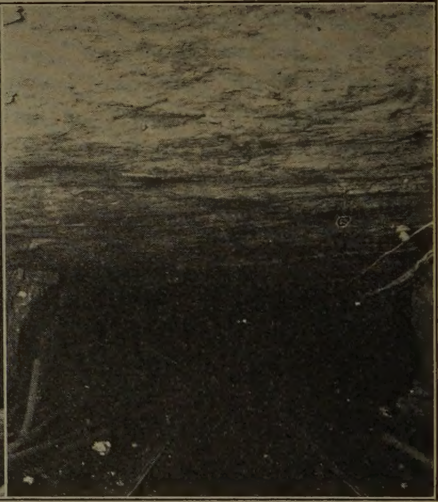

Fig. 2.—Shale roof in an entry that has always been ventilated with cooled air.

cold air in winter and warm air in summer cause the roof to contract and expand alternately, and therefore to loosen and fall.

The following observations at Saxton mine led to the assumption that heat and not moisture was the cause in this particular case. It was noticed that fresh falls never had any perceptible moisture along the bedding planes where they broke away from the roof. At one place in an entry adjacent to a dam the shale top was constantly sprayed with water for over a year and remained in perfect shape. Also, roof on the return air (with its more constant temperature) was undeniably better than roof on the intake side (varying air temperature).

Air Cooling Adopted

Consequently, it was felt that if a method could be devised to keep intake air at the normal mine temperature the roof would be considerably

improved. A study was made of the various air-cooling methods in use, with due regard to low installation and operating cost. It was found that the direct water spray type was best suited to conditions at Saxton and could be built at relatively small cost.

In common with other Indiana and Illinois mines, haulage is on the return air and a blowing fan is used. The fan is located at the main

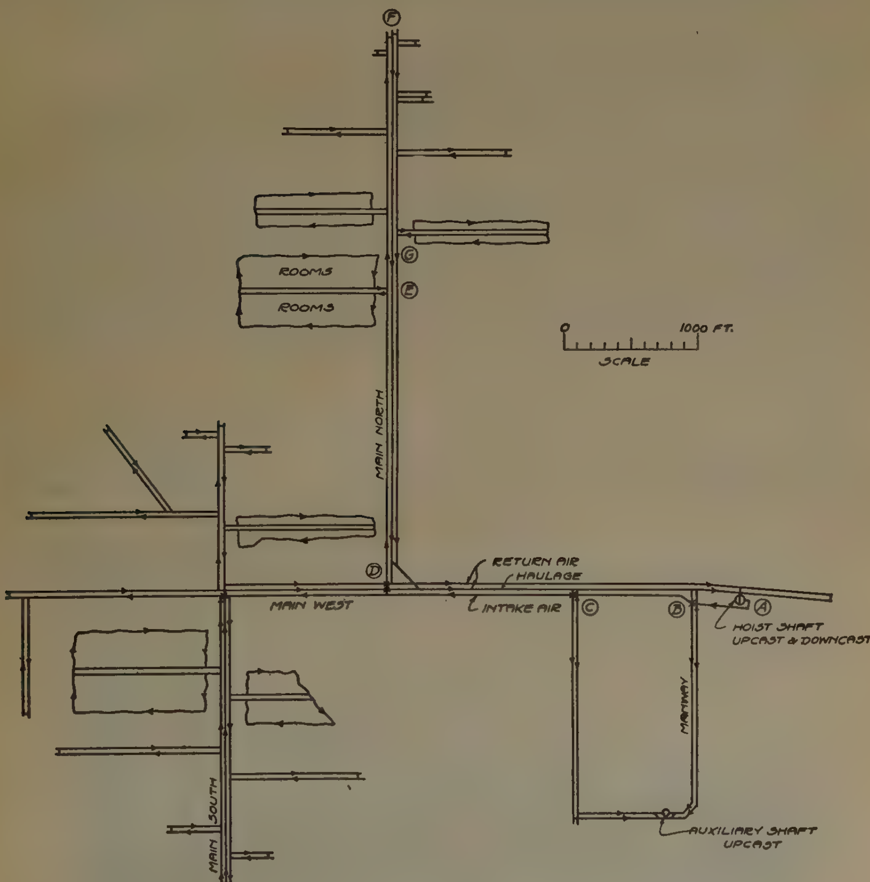

FIG. 3.—COURSE OF VENTILATING CURRENTS THROUGH MINE.

shaft, which is divided into downcast and upcast compartments with the skip hoist and emergency stairway in the upcast. Part of the return air splits (Fig. 3) and makes its exit through the auxiliary (man and material) shaft, which for various good reasons is 2100 ft. from the hoist shaft.

The cooling plant could have been located at any one of three general points: on the auxiliary shaft bottom, near the foot of the main shaft, or on top at the fan. Location on the bottom was found to cost more,

on account of long pipe lines and pumping charges, and had other disadvantages, so the surface position was chosen, where supervision would be easier and adjustments could be made at any time without entering the mine. However, a surface plant is less efficient from a cooling standpoint, as will be shown later.

It had been observed that sump water in the mine was nearly constant between 56° and 58° and that return air at the upcast shafts also was 58°, except during hot weather, when it rose to about 65°. Roof shale had been noted to remain in good shape where the air temperature did not exceed 60° F. by more than a degree or so.

The cooling problem therefore resolved itself into creating an air temperature as close to 58° as would be economical with cooling water at 56° F. Calculations were then made to determine the amount of water necessary to cool the intake air as near 58° as feasible under the maximum temperature and humidity conditions likely to be encountered. It was found that a temperature of 61° to 63° F. could be maintained on the hottest, most humid days without consuming an excessive amount of water—a temperature which was deemed satisfactory from a roof standpoint.

Description of Cooling Plant

As illustrated by Figs. 4, 5 and 6, the air-cooling plant consists primarily of four sets of water sprays through which all air passes before

Fig. 4.—Cooling plant with engine rooms and fan in background.

entering the mine fan. The spray-heads are of a patented, self-flushing type and the sets work in pairs—two always on fresh water and the other two either on fresh or recirculated water, depending on the temperature of the outside air.

Water issues in a fine sheet of spray, filling the chamber with a mist, each small particle of which comes into intimate contact with the

air, thereby affording an efficient heat exchange. Zigzag baffle plates set vertically beyond the innermost set of sprays prevent entrained moisture from being carried through the fan and down the shaft.

On either side of the spray chamber are by-passes which are opened in winter to lessen air friction. In one of these are located the recirculating pump, piping and automatic temperature-control apparatus. The "cooler" is substantially housed in a brick and concrete structure, similar in design and forming a unit with the fan housing.

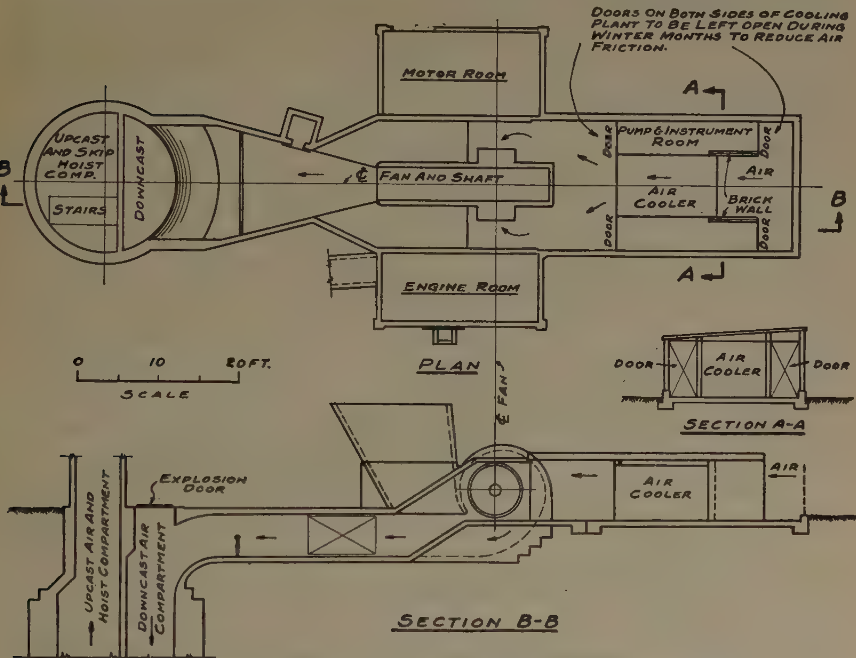

FIG. 5.—LOCATION OF COOLING PLANT WITH REFERENCE TO MINE FAN AND SHAFT.

Cool water is obtained from deep wells that supply the mine plant, a near-by village and an adjoining manufacturing plant. Used water drains away by gravity through a tile line.

Pertinent data on the cooling plant are:

Quantity air entering mine, cu. ft. per min.	40,000
Temperature inside mine	58 to 61° F.
Relative humidity in mine, per cent	97 to 100
Temperature of cooling water	56° F.
Fresh water used, gal per min.	0 to 500
(Seldom over 250 gal. per min.)	
Horsepower of recirculating pump motor	7½
Horsepower of air-compressor motor	¼
Resistance of "cooler" to air, inches water gage	7/16

Mines are ordinarily much drier in winter than in summer and the coal-dust explosion hazard consequently is increased. It was thought that at some time it might prove desirable at Saxton to add to the moisture content of intake air in winter, therefore the spray-head installed

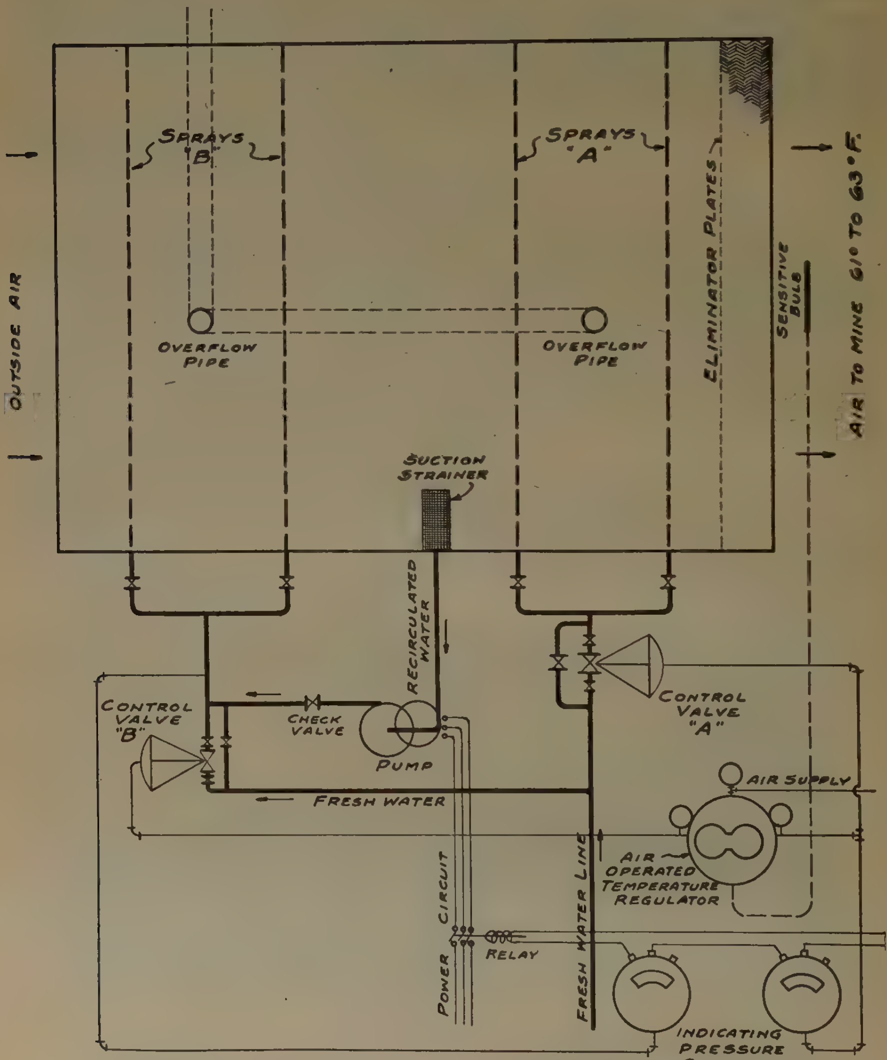

FIG. 6.—AUTOMATIC TEMPERATURE-CONTROL SYSTEM.

was of a type that could be used with steam in winter as well as with water in summer, with the same control in both cases to govern the intake temperature. However, Saxton mine has little dust and naturally is rather wet, so steam has not yet been used.

Automatic Temperature Control

From the time the cooling plant was installed in 1927 up until the latter part of the summer of 1930, the amount of water was regulated manually once a day, or less often, depending on changes in the outside temperature. Ordinarily the plant was started during the first warm spell in the spring and run continuously until frosty weather in the fall. This was necessarily inefficient, as often during a night the temperature would drop so that no cooling was needed; in fact, at times, the plant was actually heating the intake air. On the other hand, there would be hot, humid days when the intake would need more cooling than it was receiving.

Fig. 7 illustrates the wide daily temperature range at Terre Haute. The chart shows that during the warm season of 1929 there were 84 days on which it was necessary to cool the mine air for the entire 24 hr.; on 33 days no cooling whatever was needed, and on the other 97 days it was necessary to cool the air only part of the time. Because of the uncertainty and inefficiency of manual control an interesting method of automatic control was devised which, at all times, regulates the amount of water for best roof results.

When the temperature of the air entering the mine rises above 61° F. it causes a gas in the sensitive bulb (Fig. 6) to expand, and this expansion is transmitted through an air-operated temperature regulator to control valve *A*, causing it to open and admitting water to sprays *A*. At the same time an electrical contact is made which starts a motor-driven centrifugal pump. This takes the discharge water from sprays *A* and recirculates it through sprays *B* to give the air a preliminary cooling and thereby materially reduce the volume of fresh water required per minute.

If the outside temperature increases, the cooled air will exceed 61° F. and will act further on the sensitive bulb which, in turn, admits a larger volume of fresh water through valve *A* into sprays *A*; this action will continue with increasing temperatures until valve *A* is wide open, admitting all possible fresh water to sprays *A* and recirculated water to sprays *B*.

Let us suppose it is a hot, humid day and that the outside temperature rises still higher, so that the cooling condition described above is insufficient to hold the intake air to 61°. When the temperature reaches 63° its action on the sensitive bulb actuates valve *B*, admitting a small quantity of fresh water to sprays *B* mixed with the recirculating water from sprays *A*. This is possible because the pressure of the fresh-water line is 60 lb. per sq. in., and that of the recirculating water is 30 lb. per sq. in. With further increase in outside temperature (as on a very hot day) valve *B* opens wider, mixing still more fresh water until the pressure of this fresh water is about equal to that in the recirculating water line

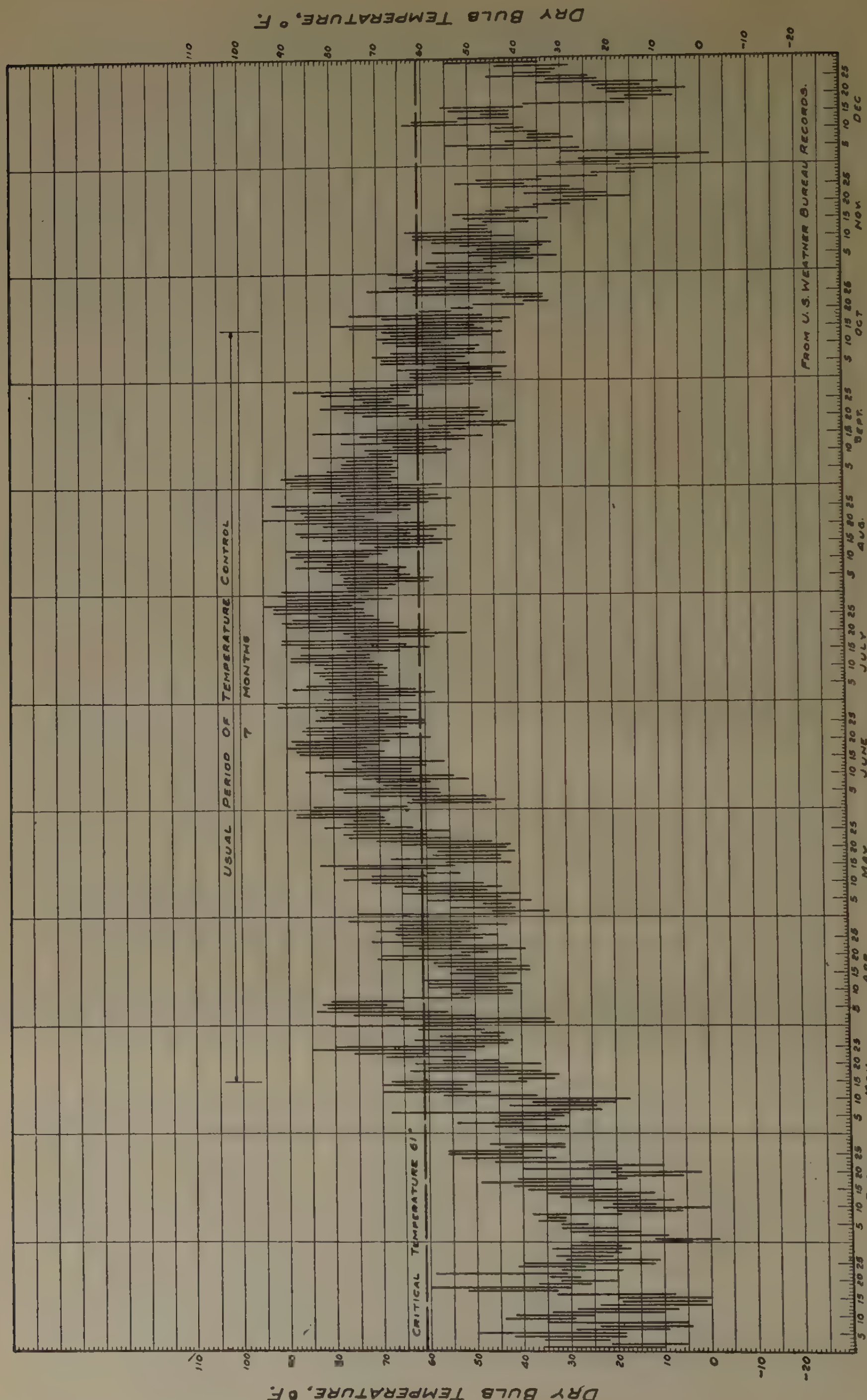

FIG. 7.—DAILY TEMPERATURE RANGE AT TERRE HAUTE, IND., DURING 1929.

(30 lb. per sq. in.). At this stage an air-operated pressure control stops the pump motor while a check valve in the recirculating water line prevents the fresh water from going in the wrong direction.

In this case, both sets of sprays are using fresh water and when both are wide open the maximum cooling effect is obtained. Only on the hottest days with a temperature of 95° or more has this been necessary. With decreasing temperatures these operations are reversed.

Results of Air Cooling

Beneficial effects on the mine roof resulting from control of the temperature of the mine air have fully met all expectations. This is illustrated best by the fact that the roof is now as good in summer as in winter. Safety is promoted, haulage is unhampered by falls, trolley wires stay up, rails are not slippery, deadwork cost is at a minimum and airways are not choked.

Throughout the mine the air is nearly saturated but despite this high humidity the temperature is low enough to fall within the comfort zone for men at work. The miners have noticed and seem to like the cool air and it is believed that labor efficiency is promoted.

Fig. 8 presents graphically the dry-bulb temperature and humidity when following the course of the air from the cooling plant through the mine workings and then back to the upcast shaft. Readings were taken at frequent intervals with a sling psychrometer.

On the day on which these readings were taken, the cooler lowered the air temperature from 83° to 62° F., while raising its relative humidity from 63 to 98 per cent. Passage through the mine fan and drift raised the temperature 4° and there was a further fraction of a degree rise in the downcast. From the foot of the shaft the air gradually cools to a nearly constant minimum of 58° F.

Location of the cooling system on the bottom would eliminate heating caused by the fan and shaft; in fact, the shaft would have somewhat precooled the air. However, other considerations, as enumerated, determined its location on top.

Costs and Savings

Cost of installing this particular air-cooling apparatus for the conditions found at Saxton mine is as follows, including labor, material and freight:

Air cooler	$2,107.00
Pipe lines, pump, etc	907.00
Power wiring	117.00
Concrete and brick housing	1,173.00
Automatic control apparatus	500.00
	$4,804.00

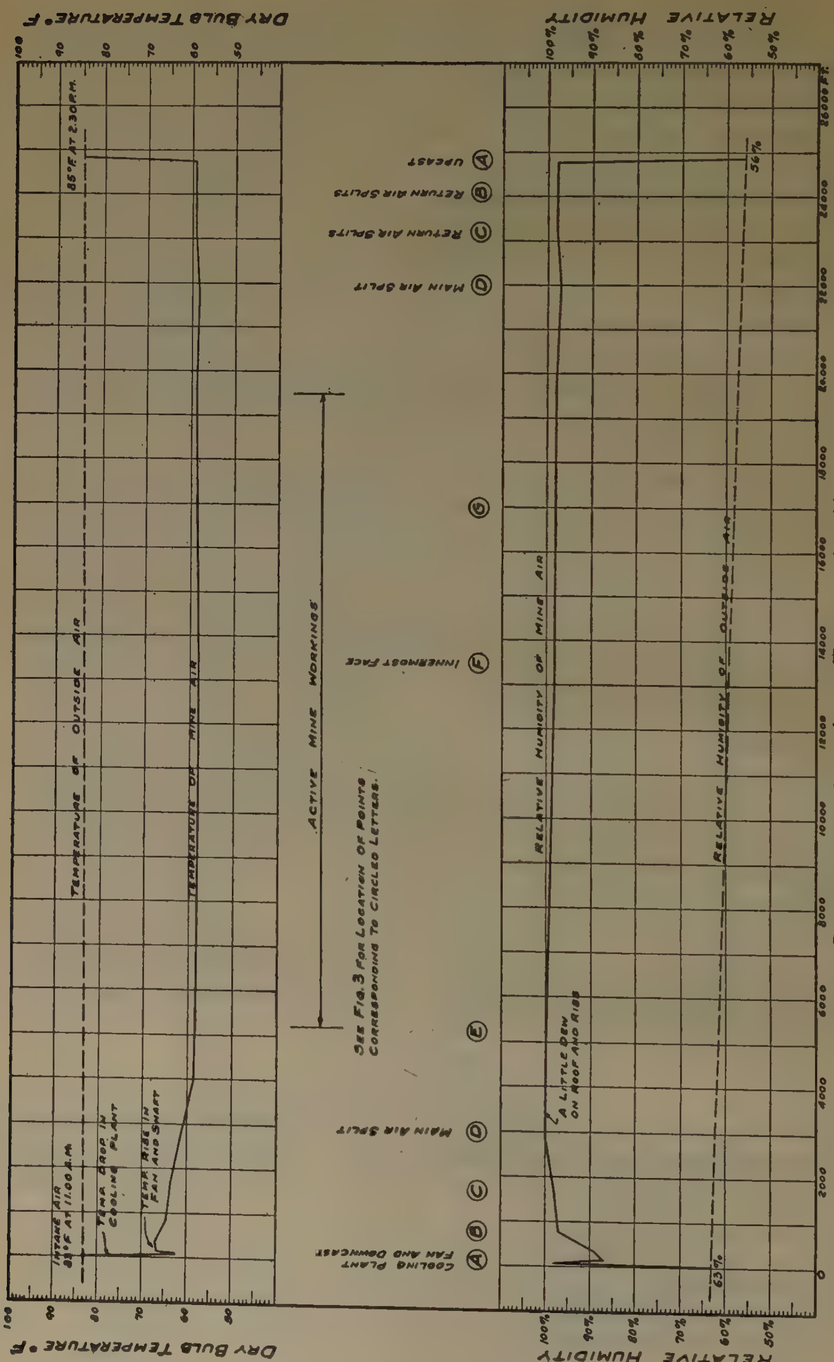

FIG. 8.—TEMPERATURE AND HUMIDITY READINGS TAKEN IN SAXTON MINE.
See Fig. 3 for location of points corresponding to circled letters.

The cost of cooling air balanced against savings resulting from less timbering, slate loading, haulage delays, torn down trolley lines, etc., would be interesting, but is not possible to ascertain because mining conditions were changed considerably when the cooling plant was installed and the tonnage output more than doubled. Therefore the only way to make a comparison would be not to operate the cooler at all for one summer, which, of course, is out of the question.

However, the following estimates give the cost of operation compared to certain savings that are conceded by those familiar with conditions at Saxton:

Estimate of Operating Cost of Air Cooler

(Based on 7 Months per Annum)

	Cost per Annum
Recirculating pump, air compressor, water supply, power cost	$1,500.00
Higher water gage, due to resistance of sprays, power cost...	220.00
Attendance labor and repairs...........................	75.00
Amortization of investment...........................	585.00
	$2,380.00

Cost per ton of coal (450,000) = $0.0053

Estimate of Savings Due to Air Cooling

At least four timbermen and slate cleaners not needed during the warm months................................	$4,000.00
Safety..	Intangible
Increased production................................	Intangible
Increased efficiency of men due to cool air................	Intangible
Less excess moisture on rails and equipment in summer....	Intangible
Timbering and ties more resistant to decay...............	Intangible

As far as the writers of this paper know, there is only one other coal mine besides Saxton that has used conditioned air to prevent roof falls; that is the mine of the Hegeler Zinc Co. near Danville, Ill., where a "dehumidifier" was installed in the belief that excess moisture caused the roof to fall at this particular operation.

DISCUSSION

(*A. C. Callen presiding*)

G. S. Rice, Washington, D.C., opened the discussion by agreeing that uniform conditions of temperature and humidity undoubtedly reduce roof falls, but he remonstrated against using the main return air course as a haulage road, as is the practice in Indiana mines and as necessitated by the plan proposed in the paper. While Indiana mines are called nongassy, he pointed to the frequency of explosions as proving them actually to be gassy. As to roof maintenance in the main intake air course, he cited the experimental mine of the U. S. Bureau of Mines, where the roof in certain portions of the main entries has been gunited since 1913 and is holding up

excellently with occasional reguniting in some places, the need of repairing being largely due to the heat of over 1200 explosion tests, whereas in other mines of like age in the district the entry roof usually cuts up very high. He urged the use of gunite for this purpose, both for safety and economy. Another practice he decried is putting the return and intake compartments in the same shaft, as the results may be, and in some cases have been, disastrous when explosions have wrecked the partition between them.

T. T. Read, New York, N.Y., pointed out that fortunately some elements of the situation were favorable; for example, if the normal rock temperature had been much above or below 60° the solution would have been much more difficult. At St. John del Rey, for instance, the top men complain about the low temperature of the cooled air, so not everyone is pleased with the results even of that installation. The authors' installation is a creditable piece of engineering work.

D. Harrington, Washington, D.C., warned against generalizing as to the relative effects of variable temperature or variable humidity on roof, saying that in some cases the difficulty apparently arose from one and in some from the other. He cited western mines in which large volumes of air, up to 200,000 cu. ft. per min., have been passed for years through long intake entries with shale roof with annual extremes in temperature from 30° to 100° F. or more, yet with no serious cutting or sloughing. He attributes this excellent roof behavior to the fact that there is little or no "sweating," because of the low humidity of the air in those regions. A. C. Callen, Urbana, Ill., asked about the daily range in temperature and Mr. Harrington said it might be wide, say from −30° to 30° or 40°.

E. McAuliffe, Omaha, Neb., said that in some Wyoming mines that formerly were ventilated by fans on the outcrop, it had become very costly to maintain the return air course properly, the falls sometimes reaching staggering proportions, so as to restrict the volume and raise the water gage. Since shafts have been sunk and the former returns used as intakes, they have dried up and the roof has become hard, much like concrete. The result is that falls have practically ceased, with an improvement in ventilation, reduced timbering costs, and so forth.

W. W. Lynch, New York, N.Y., reported that where feasible at the United Verde mine the air is conditioned to stop sloughing. In places too difficult to condition, guniting is used effectively. An extreme instance of this was a stope back about 125 ft. long by 50 ft. wide, which was gunited so that men might work under it with safety.

A. W. Hesse, Nemacolin, Pa., asked if Indiana mines are required to rock-dust with the same degree of efficiency as is required in Pennsylvania mines, where the action of humid air and the slacking of rock dust must be considered. This condition becomes so pronounced in summer that it is almost impossible to sample the rock dust distributed in the various parts of the mine. He has been investigating the use of gunite, and in discussing the matter with the Gunite Construction Co. of Fairmont, W. Va., received the estimated cost at 12¢ per square foot for this form of construction. During the investigation, he learned that the Consolidation Coal Co. had had some experience with guniting, for the purpose of preventing roof falls, and that this gunite had been put on for a thickness of from 1 to 1½ in. in order to obtain satisfactory results. Mr. Hesse asked Mr. F. F. Jorgenson, of Fairmont, W. Va., if he would relate the experience with guniting at the mines of the Consolidation Coal Co.

Mr. Jorgenson replied that they have done a great deal of guniting, but in only one place near the shaft bottom, where a large storage-battery charging station at mine No. 86 was gunited. They never use a thickness less than 1 in. and have found

it necessary to remove all loose material from the surface to be gunited. This must be done with great care. The total cost runs approximately 18¢ per square foot. They recently completed 300 to 400 ft. of very bad roof which had fallen in in places as much as 15 ft. soon after the entry was driven, timbers having failed to hold. Since guniting there have been no falls. Some of the entries first gunited have stood almost intact for two years.

Mr. Rice expressed satisfaction with this experience as he had suggested the guniting in mines in a paper in 1911 and after trial in the experimental mine had urged its use in metal as well as coal mines. Some may remember the first public demonstration of its use, when the A. I. M. E. visited the experimental mine in 1914. He said that guniting should be done as soon as possible after the entries are driven, to prevent the beginning of oxidation. He wished to make it clear that he regarded the method of roof control described in the paper as an ingenious one and that the authors are to be congratulated on their solution, in spite of the lack of safety of the general situation.

J. W. Paul, Pittsburgh, Pa., said, in connection with rock-dusting the Indiana mines, that Fig. 8 of the paper shows the relative humidity of the air to be less than 100 per cent. after it had passed the 3000-ft. level from the intake shaft, so that from that point there would be no condensation of moisture on the walls.

F. F. Jorgenson, in commenting on the wetting of rock dust, said that in some mines which admit steam into the intake the walls become wet for some distance and that the rock dust sets into a solid cake on drying.

B. F. Tillson, Montclair, N. J., described a unique method of fighting a stubborn and almost inaccessible mine fire. Finding themselves unable to cope with the fire by their usual methods, the workers equipped the usual low-pressure atomizing spray nozzle to spray water under pressure of 250 lb. per sq. in. The ventilating fan was then arranged to blow air into the fire zone, first passing it through the atomized moisture from the sprays, relying on the entrained moisture to increase the specific heat of the air and blanket the fire. Although it seems strange to blow air on to a fire, apparently this method succeeded, as what threatened to be a disastrous fire was brought under control. Mr. Tillson spoke of the stubbornness of mine fires, saying that his experience had been that it was almost impossible to extinguish smoldering timber fires without complete immersion in water.

O. A. Glaeser, Jerome, Ariz. (written discussion).—An ore-haulage tunnel, some 6500 ft. long, was for years a part of our intake air system. As such it gave no trouble. The tunnel was driven through almost horizontally bedded limestone which contains thin seams of clay. Several years ago the intake system was changed, resulting in a slightly outcast condition in this haulage tunnel. The air passing through it was leakage air from the return, and therefore practically saturated and warmer than the rock temperature in the tunnel. Within a few weeks, the roof began to slough, and shortly thereafter there were roof falls of considerable proportions. The matter had to be corrected at once as the haulage of ore was interrupted frequently.

Fresh air direct from the fan was introduced into the tunnel near the inside end. At first 25,000 cu. ft. per min. was turned into the tunnel; later this was reduced to about 5000. A crew was sent through the tunnel to bar down all loose rock and clean up. The tunnel was dry within six days. Since that date—about three years ago—there has been no trouble from falls of roof.

The temperature of the air now passing through the tunnel is lower than the rock temperature, except during July and August, when it is slightly warmer. However,

the humidity is extremely low, approximately 20 to 30 per cent., so that no moisture is precipitated from the air.

J. W. Paul (written discussion).—When there was much discussion in 1908 as to the methods suitable for adding humidity to mine air during the winter months, to allay the coal dust and prevent mines from drying out, a former banker in Charleston, W. Va., suggested that canvas might be suspended at different parts of the mine on the intake air current and kept wet by water in perforated pipes. About the same time spraying nozzles were being tried out along the intake air currents of a number of mines for the purpose of adding moisture to the intake air, and, prior to this, steam had been used for the same purpose in the winter months. This spraying was intended to prevent the drying out of the mine and to reduce the coal-dust explosion hazard, but the coal dust would not take up sufficient moisture to render it nonexplosive when blown into the air. At a number of mines where steam was admitted to the intake air current at irregular intervals, much destruction of the roof resulted, not as the result of moisture but as the result of sudden changes of temperature.

During the years 1908–09, Carl Scholz[1] began a study of conditioning the air in a coal mine in Oklahoma, to determine the effect of temperature and moisture on the roof, as he had observed the frequency of roof falls during certain seasons of the year when there were extreme changes in the temperature. Mr. Scholz said that apparently the contact of the air with the water tended to reduce the number of dust explosions without increasing the number of roof falls. The absence of effect on the roof was attributed to the uniformity of the humid condition of the mine air, in contrast to the former extreme changes from winter to summer. Frank Haas[1] gives details of the conditioning of mine air on a large scale at a coal mine in West Virginia but as the conditioning was for the purpose of the prevention of drying of the coal dust, no comments were added on the effect on the mine roof. However, he says: "The fact demonstrated by these observations is that radiation from the side walls has considerable effect in raising the temperature of the air." These observations were made during a cold month when the intake current ranged from 22° to 37° F. and in traveling 4000 ft. along the entries the air had assumed a temperature of 50°, whereas the return air current during the coldest weather was 55°. It may be seen that air traveling at a velocity not exceeding 500 ft. per min. will practically assume normal mine temperature in a distance of 4000 ft. in a climate like that common in northern West Virginia.

The converse of this exchange of temperature should be expected during the hot or summer months, and here we will have warm air entering the mine and losing its temperature through contact with the walls of the mine. Therefore, in traveling a distance of 4000 ft. the warm summer air should be reduced to mine temperature. The intake air current will have been reduced to the dew point before traveling this distance, and much of its moisture precipitated, through loss of temperature which has been taken up by the walls of the mine, particularly the roof. To protect the roof from this change in temperature and the accumulation of moisture, each of which has a detrimental effect on mine roof, particularly shale and clayey shale roof material, some conditioning of the air will be beneficial, and the paper by Messrs. Fletcher and Cassidy has shown how this may be accomplished in a particular mine in the State of Indiana. It is somewhat surprising that the intake air can be reduced by such a small quantity of water. It is noted, however, that the temperature of the air has been reduced to a point where the air throughout the mine beyond the first 3000 ft. is slightly below full saturation and the temperature fluctuates through only a few degrees. Reference is made to the presence of lime in the roof and it is at least sug-

[1] Chapter in The Explosibility of Coal Dust, U. S. Bur. Mines *Bull.* 20 (1911).

gested that moisture causes the lime to swell and thus aid in the disintegration of the roof material. It is questionable whether the lime appears in any form other than calcium carbonate ($CaCO_3$), which is relatively insoluble in water and does not possess the ability to take up water and swell to a new dimension. In some parts of Ohio and Illinois a limestone forms the roof over the coal and it does not show signs of disintegration upon exposure to moist air over long periods of time.

Shale that is intermittently wet and dry disintegrates more rapidly than when kept wet or dry. There is a certain amount of expansion when it is made wet and a contraction when it dries; moisture may act in the same manner as an increase in temperature, and when both act at the same time the action is increased.

At the Bureau of Mines experimental mine near Pittsburgh the mine air has been conditioned by the use of water which reduced the temperature of the intake air below its dew point and the test zone kept dry during the summer months. The writer has recommended conditioning of mine air as a means of preventing falls of roof during the summer months.[3]

The plan described in the paper by Fletcher and Cassidy is practical evidence of the efficiency of the scheme and is well adapted to the mines where the course of the ventilation is as indicated in the paper, return air being on the main haulage. Many of the coal mines developed in the Appalachian field, and most all gassy mines, confine the intake current to the haulage roads and it is not practicable to install air conditioning in the haulage roads. Should the main intake air current be through an auxiliary shaft and passed through a series of rooms or headings before being admitted to the main haulage, it would be feasible to install the air-cooling devices and appliances. However, few mines are developed with this a part of the scheme. To install air conditioning in these mines would involve the use of doors somewhat near the shaft bottom, and these would materially interfere with the haulage.

It is gratifying to know that a scheme for humidifying mine air found ineffective in allaying coal dust in cold months is now proved efficient for dehumidifying mine air in the hot months, all by the use of water; also, that water in the form of a spray may be effective in the control of the temperature of mine air within certain ranges above freezing.

H. I. Smith, Washington, D. C. (written discussion*).—The paper by Fletcher and Cassidy is one of the most valuable papers in conjunction with the mining of coal in Indiana that I have read. Broad application of the conclusions may be expected. My experience in Indiana is that in many mines it is very expensive to keep the air courses open through the annual seasonal changes. This atmospheric condition is not confined to areas close to the intake of the coal mines—the roof often may be seriously affected over long distances from the intake. It is not unusual for a cutting of the roof to run 100 ft. or more along the entry within a few minutes, breaking the crossbars as it goes. Roof cutting is also bad at other places, particularly in Oklahoma, and there also it is a big factor in the cost of production and life of the mines. A number of attempts have been made during the past 20 years to find a method for keeping the roof from spalling and cutting, but none were followed to a consistent conclusion. One of the most recent attempts is that of the McAlester Edwards Coal Co. of Pittsburg, Okla. W. W. Fleming, district mining supervisor, gives the following information in regard to an indirect method of air conditioning for roof control:

While the roof is hard and strong in this mine, it is extremely susceptible to moisture and, during the summer months and high humidity, "sweats," using a local

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[3] J. W. Paul: Falls of Roof in Bituminous Coal Mines. Influence of the Seasons and Rate of Production. U. S. Bur. Mines *Tech. Paper* 410 (1928).

* Published by permission of the Director, U. S. Geological Survey.

term, which causes the roof to break and cut. This is the only trouble of any moment encountered by roof falls. For instance, on the morning of May 16 there was a heavy rain followed almost immediately by bright sunshine. Before the shift was over on this date, in one entry alone the roof was loosened to a dangerous condition for a distance in one stretch of 560 ft., and four rooms on this entry caved badly. The roof in the entire mine was affected to some extent.

In order to preserve the roof, this being the slack season, it was decided to shut off the ventilation from all but the development work in the 5 East and 5 West entries. This was accomplished by placing temporary brattices and deflecting the main air current from the inner and long entries in the mine. Just enough air is used to make the development work safe and comfortable, and as the amount of precipitation is directly proportional to the amount of air entering the mine, the "sweat" is being held down practically to nothing and the mine roof is not being damaged.

In Alabama the roof in a Federal coal lease cut to such an extent that it became almost impossible to keep the mine open even with a most generous expenditure for men and timber. The roof had cut about 7 ft. back over the coal on both sides of the bottom parting, and about 6 ft. high. The cutting was seasonal, and as in Oklahoma, when the shale dried out and then took on moisture it expanded and spalled. This mine was saved by the use of gunite and the number of timbermen was reduced from 7 to 1. The one timberman was retained to pull down loose sections in the area that had been gunited and replace them with fresh gunite. The success of the control of the roof was in prompt repair of any loose section before the moisture could affect the shale. These observations indicate that either a wet or dry roof would stand in these mines but when the moisture was alternately removed and deposited by means of the air current the roof would fail. There is room for further report on this question, to determine whether uniform humidity or uniform temperature, or both, is the controlling factor and possibly laboratory tests may reveal the effect of each. My belief is that variation in moisture and the capillarity of the shales are the more important variables.

J. H. Fletcher and S. M. Cassidy (written discussion).—Although the particular air-conditioning system described is at a mine that already had a blowing fan, with haulage on return air, there is no reason to prevent its application at mines using an exhaust fan with haulage on the intake. If such a mine has two intake openings, the matter is simplified, but if there is only one intake it is necessary to by-pass the air through the conditioning apparatus and provide an air-lock on the haulage entry. This means doors—but there is no reason nowadays why these doors cannot be arranged to work both automatically and safely without interference with fast haulage.

The points raised by Mr. Rice regarding potential dangers of haulage on the return air and double-compartment air shafts are hardly pertinent to the topic under discussion (however true they may be), as the cooling plant was installed at an operating mine and it was necessary to meet existing conditions.

Question has been raised at various times as to the efficacy of air-cooling after a mine has developed to some distance from the shafts, the argument being that intake air gradually assumes the uniform mine temperature and that the benefits of conditioned air are applicable only to a section of entry near the downcast. Experience at Saxton mine proves that while incoming air does eventually reach the normal mine temperature, the distance required is much greater than usually thought and the bad effects of summer air continue to a lower temperature than might be supposed.

In 1926, before the conditioning plant was installed at Saxton, the total length of air travel was 18,200 ft. During the summer, the roof over this entire distance was cutting and falling, in rooms as well as entries, and it was necessary for all motormen to be provided with sledges and shovels to clear the track of the numerous and frequent

slate falls. Temperature of air near the upcast registered 65° F. on warm days. The critical temperature at Saxton is believed to be around 63°, as roof never gives trouble until a degree or so higher.

Mr. Smith confirms the serious effect of atmospheric conditions on roof over long distances from the intake, based on his experience in Indiana and elsewhere.

Slacking of rock dust in summer at Nemacolin mine, due to humid air, as mentioned by Mr. Hesse, can be eliminated with the type of air-conditioning apparatus described in our paper, because it dehumidifies as well as cools. Even the slight dew on the roof for the first 3000 ft. of intake at Saxton could have been eliminated by placing the cooling plant at the bottom of the downcast.

Table 1.—*Analysis of Samples of Gray Slate Roof at Saxton*

	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe, Per Cent.	Fe_2O_3, Per Cent.	Ignition Loss, Per Cent.	CaO,[a] Per Cent.	MgO,[a] Per Cent.
Sample 1, north side of mine................	58.67	20.90	5.88	8.40	9.25	0.40	1.81
Sample 2, south side of mine................	58.30	21.40	5.25	7.50	8.75	0.38	1.81

[a] In silicate form.

Table 2.—*Readings on Cold Day at Saxton Mine*

Location on Fig. 3	Cumulative Distance of Air Travel	Temperature (Dry Bulb), Deg. F.	Relative Humidity, Per Cent.
A	On top	15	69
A	Foot of shaft	25.5	51
B	800 ft.	31	66
C	1,600 ft.	34.75	73
D	3,000 ft.	45.5	85
E	5,200 ft.	50.75	85
F	13,500 ft.	57.5	94
	Return Air		
G	17,100	57.5	94
D	21,800	56.75	97
C	23,200	56.5	99
A	24,800 (foot of upcast)	56.25	99

Guniting, if applied correctly, undoubtedly is effective in preserving roof, but is extremely expensive. If it had been applied at Saxton during 1926 for the entire 18,200 ft. seriously affected by summer air, the cost would have been around $30,000, even when estimated at the low figure of 12¢ per square foot. This does not include rooms which were also badly affected. At the present rate of development, if gunite were applied only on main entries, the cost at Saxton would exceed $8000 annually.

Table 1 gives analyses of samples of the gray slate roof at Saxton. The chemist states that in a qualitative test he found no carbonate but that there might possibly be a trace.

During the past winter readings were taken through the mine on several cold days and results on one of these days are given in Table 2, as a comparison with Fig. 8, the air condition in summer. The mine is naturally wet and the tabulation shows the speed with which moisture is taken up by the air.

Subsidence and Ground Movement in a Limestone Mine Caused by Longwall Mining in a Coal Bed Below

By R. Laird Auchmuty,* Pittsburgh, Pa.

(New York Meeting, February, 1931)

Foreword

The A. I. M. E. Subcommittee on Bituminous Mining has been trying for several years to secure the information that was collected by the Marquette Cement Manufacturing Co. on the subsidence of its property caused by mining coal beneath it. The Marquette company was willing to have these data used, but it was impossible to obtain them because the records had been filed away. About two years ago these records became available and the engineering data presented herewith were compiled from them by R. L. Auchmuty, assisted by L. E. Young, who was connected with the collection of the information.

On account of the cost, it is impossible to reproduce all of the data here, although the original information is on file with the records of the Committee on Ground Movement and Subsidence. The accompanying charts show representative and average data and give only the engineering information with enough description to tell how it was obtained. It is hoped that all of those connected with the original case will participate in the discussion that this will bring forth.

Howard N. Eavenson, *Chairman*,
Subcommittee on Bituminous Coal Mining.

Probably the most complete records of subsidence due to coal mining in the United States are those compiled by the Marquette Cement Manufacturing Co. at LaSalle, Ill. These records, covering a period of more than three years, were compiled and used as evidence during the suit of the Marquette Cement Manufacturing Co. vs. the Oglesby Coal Co. at Oglesby, LaSalle County, Ill., in which the cement company successfully endeavored to stop the coal company from mining under its property.

Fig. 1 shows a typical section of the geological formation. The cement company was mining the LaSalle limestone which lies about 125 ft. under the surface. The limestone is 30 to 35 ft. thick and underlain by about 15 ft. of shale. The cement company was mining about 20 ft. of the limestone, and in some places 10 to 12 ft. of the underlying shale, by the room and pillar method. The mining of the limestone always preceded the mining of the shale.

The coal company was mining by longwall advance methods the No. 2 seam of coal 42 in. thick underlying the limestone bed by 435 to 470 ft. The coal mine, a very old one, had largely mined out its original property

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* Mining Engineer, Eavenson, Alford & Hicks.

and under the limestone area being mined part was originally owned by the cement company and part by the coal company; they arranged to sever the ownership by having the coal rights in the cement company's land conveyed to the coal company, the cement company retaining all the rest, and, where the coal company was the owner, by having the fee conveyed to the cement company, reserving the coal and the right of removal. Deeds from cement company to coal company described the No. 2 coal seam and so much of the rock, clay and other minerals just

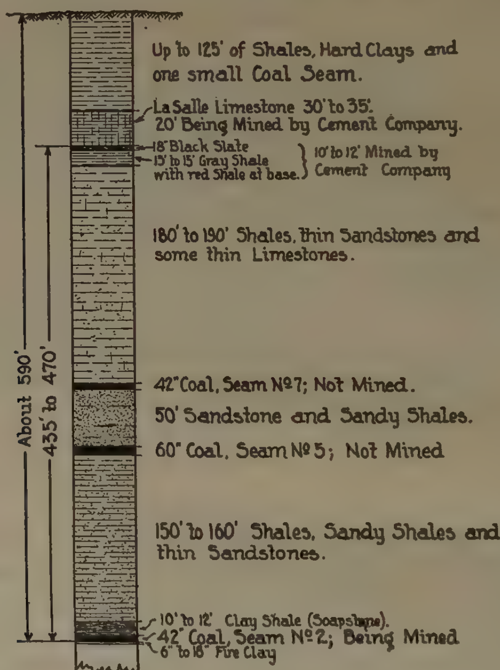

FIG. 1.—TYPICAL SECTION OF GEOLOGICAL FORMATION.

above and just below the vein of coal as might be required in connection with the mining and removing of the coal, together with the right to mine and remove the same, and the adjacent rock, clay and other minerals, "without entering upon or injuring the surface thereof," excepting the coal under the cement company's manufacturing plant. Deeds made by coal company to cement company excepted the coal and minerals just above and below, with the same right to mine and remove it "without entering upon or injuring the surface thereof." This condition, of course, was impossible of fulfillment if the coal was mined by the longwall method, which the coal company proceeded to do.

Subsidence first affected the Illinois Central Railroad which had a protection agreement with the cement company. The railroad had been

protected by pillars left in the cement mine but when the longwall workings undermined these pillars, settlement occurred. Soon after this the longwall workings advanced under the active portions of the cement mine. The workings in the limestone caved badly; the solid stone was cracked and disturbed; and mining had to be abandoned to more or less extent in the affected areas, resulting in the loss of much stone and available development.

As a result of the unfavorable conditions brought on by the coal mining, the cement company decided to protect its property by entering suit against the coal company to restrain it from further undermining the property of the cement company.

A series of base lines and bench marks was established on the surface and in the cement mine, and observations as to vertical and both longitudinal and lateral movement were made from time to time, for a period extending over three years. In all, there were 145 surface monuments and over 300 pillar plugs and several base lines in the limestone mine workings. These monuments were relocated at intervals, the surveys being made by three independent parties and the results checked against each other. A large number of photographs were also made, showing the crushing action on the limestone pillars from time to time. In addition to the field work, elaborate laboratory experiments were carried on to show the effect of subsidence on various materials, and to determine crushing strength of pillars, and so forth.

It is impossible to publish all of the data collected, on account of expense, but for the purposes of this paper it is thought that three illustrations will be sufficient to show the ground movement that is typical of this case. Numerous other charts and survey records are available to those who are interested in making a further study of the case.

Fig. 2 shows a base line, stations 53 to 67, which parallels the property line of the Oglesby Coal Co. and the Marquette Cement Manufacturing Co. The plan shows the contours of the ground, the advance of the longwall faces and the workings of the cement mine. Survey records giving the detailed observations on the stations accompany the figure.

It will be noticed that settlement started on the surface 100 ft. to over 300 ft. in advance of the coal face, and this disturbance started over a year before the coal directly beneath was mined and that preceding settlement, a slight upthrust action, amounting in this case to as much as 0.041 ft., was recorded.

This line was laid out in February, 1915, although at that time the longwall face had advanced almost to station 56, therefore the settlement charted between stations 53 to 56 is not the total for that section of line but only the amount that took place after the line was established. Settlement was still taking place three years after the coal had been mined.

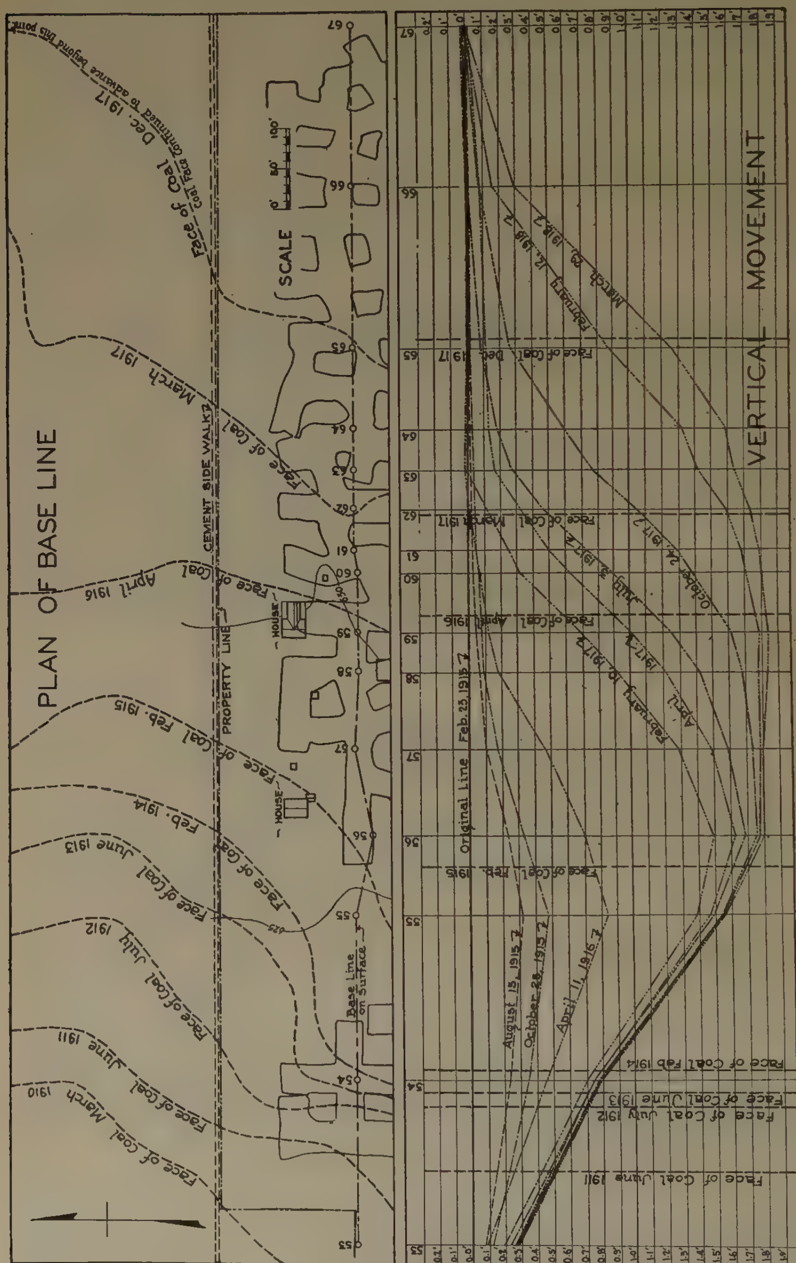

FIG. 2.—BASE LINE, STATIONS 53 TO 67, WHICH PARALLELS PROPERTY LINE OF OGLESBY COAL COMPANY AND MARQUETTE CEMENT MANUFACTURING COMPANY.

Date	53			54			55			56			57		
	Elevation	Difference Last Survey	Difference Total	Elevation	Difference Last Survey	Difference Total	Elevation	Difference Last Survey	Difference Total	Elevation	Difference Last Survey	Difference Total	Elevation	Difference Last Survey	Difference Total
Feb. 23, 1915	622.069			622.902			626.320			629.023			630.287		
Aug. 19, 1915	622.008	-.061	-.061	622.671	-.231	-.231	625.992	-.328	-.328	628.806	-.217	-.217	630.185	-.102	-.102
Oct. 25, 1915	621.949	-.059	-.120	622.558	-.113	-.344	625.846	-.146	-.474	628.700	-.106	-.323	630.112	-.073	-.175
Apr. 11, 1916	622.006	+.057	-.063	622.459	-.099	-.443	625.471	-.375	-.849	628.317	-.383	-.706	629.921	-.191	-.366
Feb. 10, 1917	621.876	-.130	-.193	622.202	-.257	-.700	624.942	-.529	-1.378	627.544	-.773	-1.479	629.021	-.900	-1.266
Apr. -, 1917	621.827	-.049	-.242	622.151	-.051	-.751	625.860	-.082	-1.460	627.411	-.133	-1.612	628.815	-.206	-1.472
Jul. 3, 1917	621.850	+.023	-.219	622.160	+.009	-.742	624.841	-.019	-1.479	627.359	-.052	-1.664	628.719	-.096	-1.568
Oct. 24, 1917	621.814	-.036	-.255	622.136	-.024	-.766	624.794	-.047	-1.526	627.277	-.082	-1.746	628.557	-.162	-1.730
Feb. 12, 1918	621.837	+.023	-.232	622.130	-.006	-.772	624.784	-.010	-1.536	627.257	-.020	-1.766	628.516	-.041	-1.771
Mar. 29, 1918	621.828	-.009	-.241	622.112	-.018	-.790	624.779	-.005	-1.541	627.230	-.027	-1.793	628.498	-.018	-1.789

Date	58			59			60			61			62		
Feb. 23, 1915	630.559			631.702			632.248			632.362			632.148		
Aug. 19, 1915	630.506	-.053	-.053	631.662	-.040	-.040	532.224	-.024	- .024	632.338	-.024	-.024	632.121	-.027	-.027
Oct. 25, 1915	630.482	-.024	-.077	631.629	-.033	-.073	632.200	-.024	-.048	632.319	-.019	-.043	632.113	-.008	-.035
Apr. 11, 1916	630.375	-.107	-.184	631.585	-.044	-.117	632.197	-.003	-.051	632.333	+.014	-.029	632.128	+.015	-.020
Feb. 10, 1917	629.644	-.731	-.915	631.014	-.571	-.688	631.924	-.273	-.324	632.119	-.214	-.243	632.039	-.089	-.109
Apr. -, 1917	629.352	-.292	-1.207	630.701	-.313	-1.001	631.629	-.295	-.619	631.864	-.255	-.498	631.815	-.224	-.333
July 3, 1917	629.154	-.198	-1.405	630.464	-.237	-1.238	631.373	-.256	-.875	631.630	-.234	-.732	631.657	-.158	-.491
Oct. 24, 1917	628.884	-.270	-1.675	630.105	-.359	-1.597	630.885	-.448	-1.363	631.095	-.535	-1.267	631.105	-.552	-1.043
Feb. 12, 1918	628.771	-.113	-1.788	629.922	-.183	-1.780	630.547	-.338	-1.701	630.691	-.404	-1.671	630.576	-.529	-1.572
Mar. 29, 1918	628.720	-.051	-1.839	629.855	-.067	-1.847	630.448	-.099	-1.800	630.578	-.113	-1.784	630.414	-.162	-1.734

Date	63			64			65			66			67		
Feb. 23, 1915	631.930			632.302			632.757			631.737			633.888		
Aug. 19, 1915	631.910	-.020	-.020	632.280	-.022	-.022	632.733	-.024	-.024	631.727	-.010	-.010	633.872	-.016	-.016
Oct. 25, 1915	631.898	-.012	-.032	632.267	-.013	-.035	632.715	-.018	-.042	631.702	-.025	-.035	633.877	+.005	-.011
Apr. 11, 1916	631.944	+.046	+.014	632.308	+.041	+.006	632.747	+.032	-.010	631.713	+.011	-.024	633.900	+.023	+.012
Feb. 10, 1917	631.929	-.015	-.001	632.309	+.001	+.007	632.719	-.028	-.038	631.729	+.016	-.008	633.929	+.029	+.041
Apr. -, 1917	631.760	-.169	-.170	632.171	-.138	-.131	632.644	-.075	-.113	631.668	-.061	-.069	633.868	-.061	-.020
July 3, 1917	631.649	-.111	-.281	632.124	-.047	-.178	632.661	+.017	-.096	631.694	+.026	-.043	633.908	+.040	+.020
Oct. 24, 1917	631.163	-.486	-.767	631.701	-.423	-.601	632.483	-.178	-.274	631.660	-.034	-.077	633.890	-.018	+.002
Feb. 12, 1918	630.524	-.639	-1.406	630.988	-.713	-1.314	631.869	-.614	-.888	671.582	-.078	-.155	633.920	+.030	+.038
Mar. 29, 1918	630.305	-.219	-1.625	630.720	-.266	-1.582	631.509	-.360	-1.248	631.435	-.147	-.302	633.888	+.032	.000

SURVEY RECORD, VERTICAL MOVEMENT, FIG. 2.

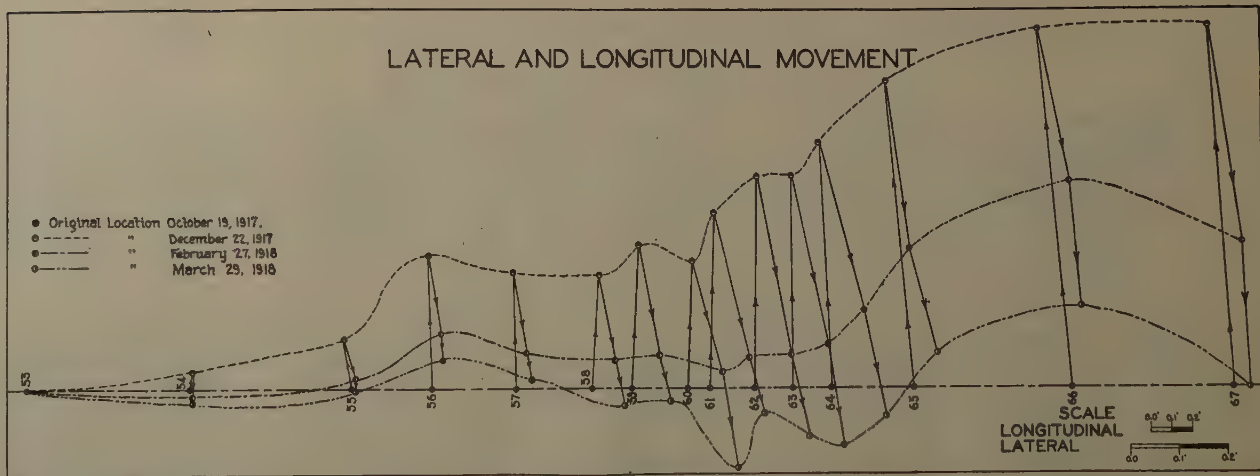

FIG. 2.—BASE LINE, STATIONS 53 TO 67, WHICH PARALLELS PROPERTY LINE OF OGLESBY COAL COMPANY AND MARQUETTE CEMENT MANUFACTURING COMPANY.

Survey Point	OCTOBER 19, 1917 Original Location North	South	DECEMBER 22, 1917 Location North	South	Movement	Direction	FEBRUARY 27, 1918 Location North	South	Movement	Direction	MARCH 29, 1918 Location North	South	Movement	Direction
53	on	line	on	line	00		on	line	00		on	line	00	
54	6-1/4"		6-5/8"		3/8"	N.	6"		5/8"	S.	5-7/8"		1/8"	S.
55		1-3/16"	on	line	1-3/16"	N.		1-1/4"	1-1/4"	S.		1"	1/4"	N.
56		19'-1-7/16"		18'-10-3/16"	3-1/4"	N.		19'-0-1/8"	1-15/16"	S.		19'-0-13/16"	11/16"	S.
57		2-13/16"	on	line	2-13/16"	N.		2"	2"	S.		2-11/16"	11/16"	S.
58		4'-6-1/2"		4'-3-3/4"	2-3/4"	N.		4'-6-3/8"	2-5/8"	S.		4'-7"	5/8"	S.
59		4'-10-3/4"		4'-7-1/4"	3-1/2"	N.		4-10"	2-3/4"	S.		4'-11-1/8"	1-1/8"	S.
60		4'-10-1/16"		4'-7"	3-1/16"	N.		4'-9-3/4"	2-3/4"	S.		5'-0-1/8"	2-3/8"	S.
61		3-15/16"	3/8"		4-5/16"	N.		3-1/4"	3-5/8"	S.		4-5/8"	1-3/8"	S.
62		6-3/4"		1-5/8"	5-1/8"	N.		6"	4-3/8"	S.		8"	2"	S.
63	on	line	5-1/8"		5-1/8"	N.	1"		4-1/8"	S.		1-1/2"	2-1/2"	S.
64		1/2"	5-1/2"		6"	N.	1-3/8"		4-1/8"	S.		1-1/4"	2-5/8"	S.
65		1'-1-7/16"		5-7/8"	7-9/16"	N.		10"	4-1/8"	S.		1'-0-5/8"	2-5/8"	S.
66	1-7/8"		10-3/4"		8-7/8"	N.	7"		3-3/4"	S.	3-7/8"		3-1/8"	S.
67	5-3/4"		14-3/4"		9"	N.	9-3/8"		5-3/8"	S.	5-3/4"		3-5/8"	S.

SURVEY RECORD, LATERAL MOVEMENT, FIG. 2.

DATE	53 to 54			53 to 55			53 to 56			53 to 57			53 to 58		
	Total Hor. Distance	Difference Last Survey	Difference Total	Total Hor. Distance	Difference Last Survey	Difference Total	Total Hor. Distance	Difference Last Survey	Difference Total	Total Hor. Distance	Difference Last Survey	Difference Total	Total Hor. Distance	Difference Last Survey	Difference Total
Oct. 19, 1917	199.823			399.264			499.417			607.186			701.546		
Dec. 22, 1917	199.831	+.008	+.008	399.222	-.042	-.042	499.400	-.017	-.017	607.176	-.010	-.010	701.581	+.035	+.035
Feb. 27, 1918	199.831	.000	+.008	399.283	+.061	+.019	499.462	+.062	+.045	607.237	+.061	+.051	701.656	+.075	+.110
March 29, 1918	199.835	+.004	+.012	399.278	-.005	+.014	499.471	+.009	+.054	607.263	+.027	+.077	701.700	+.044	+.154
DATE	53 to 59			53 to 60			53 to 61			53 to 62			53 to 63		
Oct. 19, 1917	751.278			823.232			850.641			900.932			950.399		
Dec. 22, 1917	751.324	-.046	-.046	823.276	-.044	-.044	850.687	-.043	-.043	900.958	+.026	+.026	950.396	-.003	-.003
Feb. 27, 1918	751.417	+.193	+.139	823.403	+.227	+.171	850.833	+.246	+.192	901.118	+.260	+.186	950.572	+.276	+.173
March 29, 1918	751.468	+.051	+.190	823.479	+.076	+.247	850.910	+.077	+.269	901.208	+.090	+.276	950.653	+.081	+.254
DATE	53 to 64			53 to 65			53 to 66			53 to 67					
Oct. 19, 1917	1000.650			1100.558			1300.444			1501.030					
Dec. 22, 1917	1000.600	-.050	-.050	1100.411	-.147	-.147	1300.287	-.157	-.157	1500.891	-.139	-.139			
Feb. 27, 1918	1000.800	+.200	+.150	1100.539	+.228	-.019	1300.424	+.237	-.020	1501.065	+.278	+.039			
March 29, 1918	1000.876	+.076	+.226	1100.617	+.078	+.059	1300.461	+.037	+.017	1501.110	+.041	+.080			

SURVEY RECORD, TOTAL LONGITUDINAL MOVEMENT, FIG. 2.

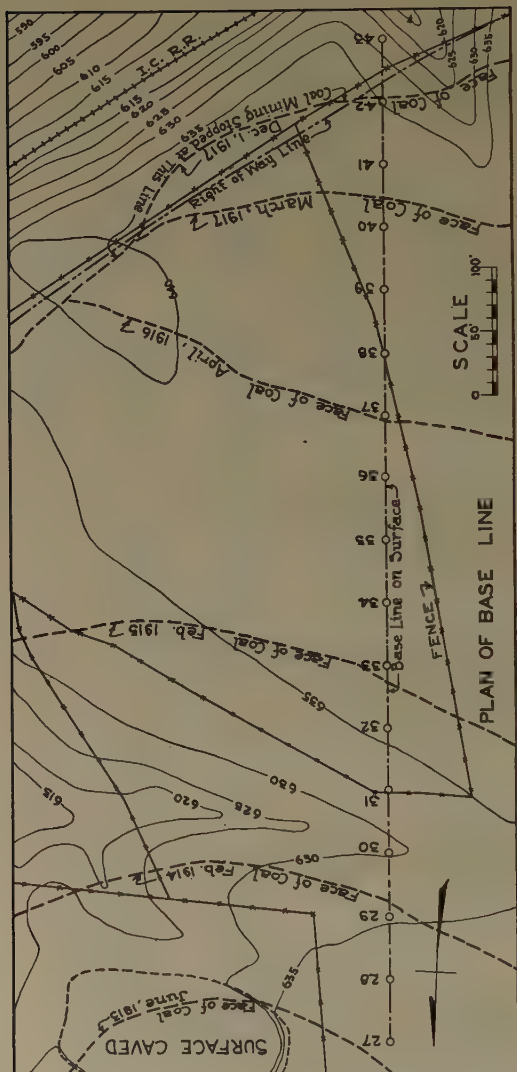

FIG. 3.—BASE LINE, STATIONS 27 TO 43.

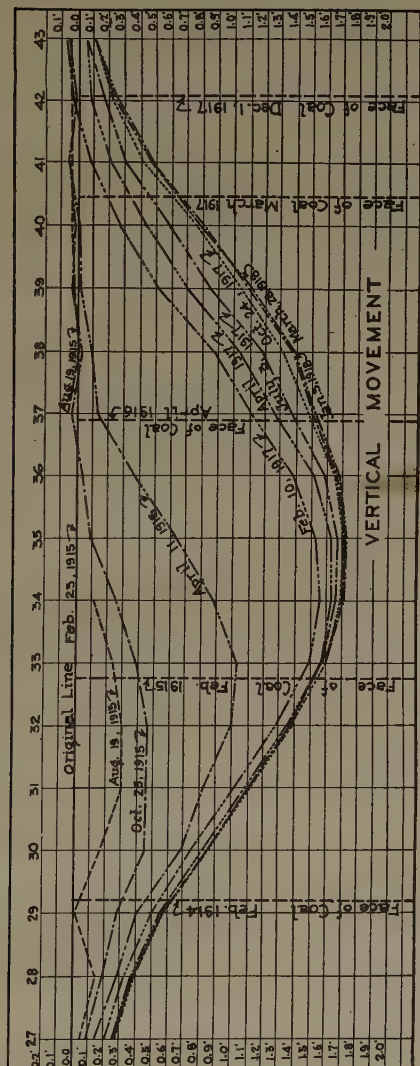

FIG. 3.—BASE LINE, STATIONS 27 TO 43.

Date	27 Elevation	27 Difference Last Survey	27 Difference Total	28 Elevation	28 Difference Last Survey	28 Difference Total	29 Elevation	29 Difference Last Survey	29 Difference Total	30 Elevation	30 Difference Last Survey	30 Difference Total	31 Elevation	31 Difference Last Survey	31 Difference Total	32 Elevation	32 Difference Last Survey	32 Difference Total
Feb. 23, 1915	638.533			637.438			635.231			631.069			635.044			638.015		
Aug. 19, 1915	638.477	-.056	-.056	637.292	-.146	-.146	635.223	-.008	-.008	630.902	-.167	-.167	634.739	-.305	-.305	637.707	-.308	-.308
Oct. 25, 1915	638.436	-.041	-.097	637.241	-.051	-.197	634.960	-.263	-.271	630.626	-.276	-.443	634.586	-.153	-.458	637.538	-.169	-.477
Apr. 11, 1916	638.395	-.041	-.138	637.160	-.081	-.278	634.820	-.140	-.411	630.384	-.242	-.685	634.220	-.366	-.824	637.005	-.533	-1.010
Feb. 10, 1917	638.331	-.064	-.202	637.107	-.053	-.331	634.733	-.187	-.498	630.317	-.067	-.752	634.027	-.193	-1.017	636.723	-.282	-1.292
Apr. -, 1917	638.266	-.065	-.267	637.040	-.067	-.398	634.662	-.071	-.569	630.243	-.074	-.826	633.953	-.074	-1.091	636.651	-.072	-1.364
Jul. 3, 1917	638.274	+.008	-.259	637.061	+.021	-.377	634.671	+.009	-.560	630.244	+.001	-.825	633.953	.000	-1.091	636.646	-.005	-1.369
Oct. 24, 1917	638.267	-.007	-.266	637.054	-.007	-.384	634.658	-.013	-.573	630.239	-.005	-.830	633.939	-.014	-1.105	636.631	-.015	-1.384
Jan. 5, 1918	638.267	.000	-.266	637.052	-.002	-.386	634.660	+.002	-.571	630.239	.000	-.830	633.935	-.004	-1.109	636.623	-.008	-1.392
Mar. 28, 1918	638.267	.000	-.266	637.053	+.001	-.385	634.661	+.001	-.570	630.236	-.003	-.833	633.938	+.003	-1.106	636.628	+.005	-1.387

Date	33 Elevation	33 Last Survey	33 Total	34 Elevation	34 Last Survey	34 Total	35 Elevation	35 Last Survey	35 Total	36 Elevation	36 Last Survey	36 Total	37 Elevation	37 Last Survey	37 Total	38 Elevation	38 Last Survey	38 Total
Feb. 23, 1915	640.176			640.951			640.582			640.488			640.996			640.511		
Aug. 19, 1915	639.937	-.239	-.239	640.837	-.114	-.114	-			-			641.047	+.051	+.051	640.501	-.010	-.010
Oct. 25, 1915	639.780	-.157	-.396	640.698	-.139	-.253	640.484	-.098	-.098	640.444	-.044	-.044	641.020	-.027	+.024	640.473	-.028	-.038
Apr. 11, 1916	639.137	-.643	-1.039	640.061	-.637	-.890	639.936	-.548	-.646	640.119	-.325	-.369	640.861	-.159	-.135	640.417	-.056	-.094
Feb. 10, 1917	638.676	-.461	-1.500	639.389	-.672	-1.562	639.055	-.881	-1.527	639.099	-1.020	-1.389	639.884	-.977	-1.112	639.640	-.777	-.871
Apr. -, 1917	638.606	-.070	-1.570	639.314	-.075	-1.637	638.954	-.101	-1.628	638.971	-.128	-1.517	639.731	-.153	-1.265	639.474	-.166	-1.037
Jul. 3, 1917	638.604	-.002	-1.572	639.288	-.026	-1.663	638.916	-.038	-1.666	638.902	-.069	-1.586	639.619	-.112	-1.377	639.331	-.143	-1.180
Oct. 24, 1917	638.575	-.029	-1.601	639.254	-.034	-1.697	638.866	-.050	-1.716	638.829	-.073	-1.659	639.512	-.107	-1.484	639.189	-.142	-1.322
Jan. 5, 1918	638.575	.000	-1.601	639.252	-.002	-1.699	638.862	-.004	-1.720	638.817	-.012	-1.671	639.489	-.023	-1.507	639.134	-.055	-1.377
Mar. 28, 1918	638.567	-.008	-1.609	639.243	-.009	-1.708	638.856	-.006	-1.726	638.806	-.011	-1.682	639.474	-.015	-1.522	639.126	-.008	-1.385

Date	[illegible] Elevation	[illegible] Last Survey	[illegible] Total	40 Elevation	40 Last Survey	40 Total	41 Elevation	41 Last Survey	41 Total	42 Elevation	42 Last Survey	42 Total	43 Elevation	43 Last Survey	43 Total
Feb. 23, 1915	639.644			638.770			637.059			634.010			629.872		
Aug. 19, 1915	639.659	+.015	+.015	638.807	+.037	+.037	637.092	+.033	+.033	634.047	+.037	+.037	629.862	-.010	-.010
Oct. 25, 1915	639.673	+.014	+.029	638.783	-.024	+.013	637.095	+.003	+.036	634.051	+.004	+.041	629.911	+.059	+.039
Apr. 11, 1916	639.625	-.048	-.019	638.756	-.027	-.014	637.101	+.006	+.042	634.020	-.030	+.010	629.916	+.005	+.044
Feb. 10, 1917	639.127	-.498	-.517	638.499	-.257	-.271	636.969	-.132	-.090	634.034	+.014	+.024	629.929	+.013	+.057
Apr. -, 1917	638.939	-.188	-.705	638.337	-.162	-.433	636.851	-.118	-.208	633.932	-.102	-.078	629.820	-.109	-.052
Jul. 3, 1917	638.789	-.150	-.855	638.181	-.156	-.589	636.757	-.094	-.302	633.832	-.100	-.178	629.817	-.003	-.055
Oct. 24, 1917	638.609	-.180	-1.035	638.005	-.176	-.765	636.635	-.122	-.424	633.783	-.049	-.227	629.761	-.056	-.111
Jan. 5, 1918	638.552	-.057	-1.092	637.966	-.039	-.804	636.588	-.047	-.471	633.768	-.015	-.242	629.762	-.001	-.110
Mar. 28, 1918	638.534	-.018	-1.110	637.989	+.023	-.761	636.573	-.015	-.486	633.760	-.008	-.250	629.758	-.004	-.114

SURVEY RECORD, VERTICAL MOVEMENT, FIG. 3.

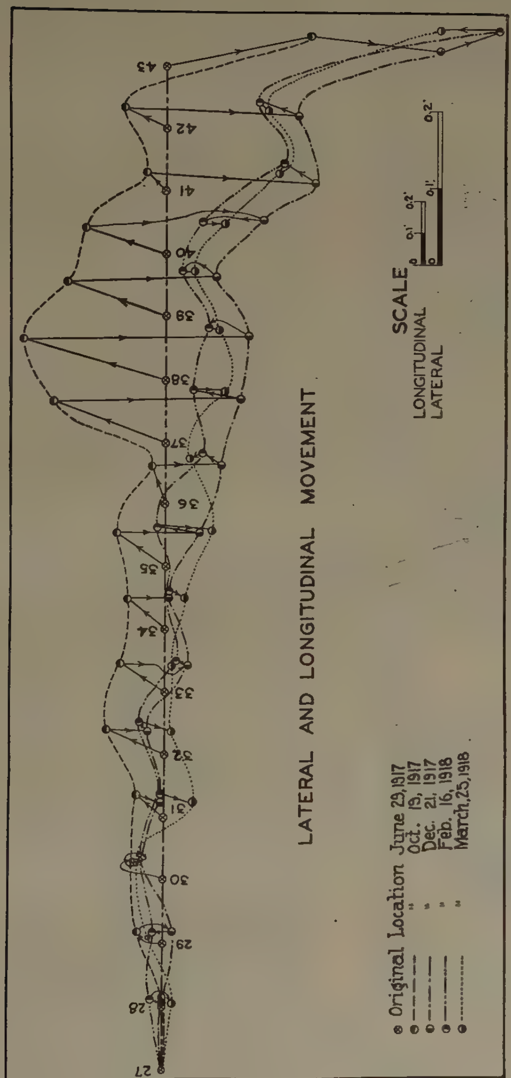

Fig. 3.—Base line, stations 27 to 43.

It should be kept in mind that mining did not stop at the last position of the coal face (December, 1917), but kept on advancing beyond the time of the last observations in March, 1918. Information as to the position of the coal face was not available after December, 1917.

Survey Point	JUNE 29, 1917 Location East	JUNE 29, 1917 Location West	OCTOBER 19, 1917 Location East	OCTOBER 19, 1917 Location West	OCTOBER 19, 1917 Movement	OCTOBER 19, 1917 Direction
27	on	line	on	line		
28		5-5/8"		5-5/8"	00	
29		5-17/32"		5-1/8"	13/32"	East
30		8-7/8"		8-3/8"	1/2"	East
31		11-1/32"		10-5/8"	13/32"	East
32		17-5/32"		16-1/4"	29/32"	East
33		14-5/32"		13-7/16"	23/32"	East
34		12-19/32"		12"	19/32"	East
35		18-27/32"		18-1/16"	25/32"	East
36		27"		26-13/16"	3/16"	East
37		22-3/16"		20-7/16"	1-3/4"	East
38		16-5/16"		14-1/16"	2-1/4"	East
39		6-31/32"		5-7/16"	1-17/32"	East
40		7-7/16"		6-3/16"	1-1/4"	East
41	on	line	1/4"		1/4"	East
42	6-19/32"		7-1/4"		21/32"	East
43	43-13/16"		41-1/2"		2-5/16"	West

Survey Point	DECEMBER 21, 1917 Location East	DECEMBER 21, 1917 Location West	DECEMBER 21, 1917 Movement	DECEMBER 21, 1917 Direction	FEBRUARY 16, 1918 Location East	FEBRUARY 16, 1918 Location West	FEBRUARY 16, 1918 Movement	FEBRUARY 16, 1918 Direction	MARCH [illegible], 1918 Location East	MARCH [illegible], 1918 Location West	MARCH [illegible], 1918 Movement	MARCH [illegible], 1918 Direction
27	on	line			on	line			on	line		
28		5-5/8"	00			5-7/16"	3/16"	East		5-3/4"	5/16"	West
29		5-5/8"	1/2"	West		5-3/8"	1/4"	East		5-3/8"	00	
30		8-1/2"	1/8"	West		8-1/2"	00			8-[illegible]/16"	1/16"	East
31		11"	3/8"	West		11"	00			11-1/2"	1/2"	West
32		16-7/8"	5/8"	West		16-3/4"	1/8"	East		17-1/4"	1/2"	West
33		14-1/2"	1-1/16"	West		14-3/8"	1/8"	East		14-1/4"	1/8"	East
34		12-5/8"	5/8"	West		12-5/8"	00			12-7/8"	1/4"	West
35		19-3/8"	1-5/16"	West		18-3/4"	5/8"	East		19-5/8"	7/8"	West
36		27-7/8"	1-1/16"	West		27-5/8"	1/4"	East		27-3/8"	1/4"	East
37		23-3/8"	2-15/16"	West		22-5/8"	3/4"	East		23-1/8"	1/2"	West
38		17-5/8"	3-9/16"	West		17"	5/8"	East		17-1/8"	1/8"	West
39		7-3/4"	2-5/16"	West		7-1/4"	1/2"	East		7-7/16"	3/16"	West
40		9"	2-3/16"	West		8"	1"	East		8-3/8"	3/8"	West
41		2-3/8"	2-5/8"	West		1-7/8"	1/2"	East		1-13/16"	1/16"	East
42	4-1/2"		2-3/4"	West	5-1/8"		5/8"	East	5"		1/8"	West
43	38-1/2"		3"	West	39-1/2"		1"	East	39-1/2"		00	

SURVEY RECORD, LATERAL MOVEMENT, FIG. 3.

The chart showing the lateral and longitudinal movement shows the oscillating character of this disturbance and numerous records of similar cases show this movement to be always of an oscillating character, rather than in one direction.

Fig. 3 shows a base line, stations 27 to 43. The observed action of this line is similar to that shown in Fig. 2. However, in this case coal mining stopped Dec. 1, 1917, in order to protect the railroad which paralleled the face at this point.

Fig. 4 shows a base line, stations A17 to A31, established in the workings of the cement mine. This line was located almost three years after coal mining had started to advance under this area, so that only a portion of the total subsidence could be observed. Much of the value of this observation was lost because the line was located too late. It appears that the action in the cement mine is similar to that observed on the surface. In this case, however, instead of settlement there was a decided upthrust in advance of the face and apparently settlement did not start until the face had advanced about 75 ft. beyond the point observed.

Fig. 5 shows the angles of draw in advance of the coal face as of April, 1916, on Fig. 2. The smaller angle is drawn to the point where subsidence is first noticeable, the larger angle is drawn to the point where slight upheaval commences. This condition is

Date	27 to 28 Total Horizontal Distance	27 to 28 Difference Last Survey	27 to 28 Difference Total	27 to 29 Total Horizontal Distance	27 to 29 Difference Last Survey	27 to 29 Difference Total	27 to 30 Total Horizontal Distance	27 to 30 Difference Last Survey	27 to 30 Difference Total	27 to 31 Total Horizontal Distance	27 to 31 Difference Last Survey	27 to 31 Difference Total	27 to 32 Total Horizontal Distance	27 to 32 Difference Last Survey	27 to 32 Difference Total
June 29, 1917	49.989			99.712			149.721			199.744			249.606		
Oct. 19, 1917	50.012	+.023	+.023	99.752	+.040	+.040	149.784	+.063	+.063	199.813	+.069	+.069	249.684	+.078	+.078
Dec. 21, 1917	49.998	-.014	+.009	99.756	+.004	+.044	149.784	.000	+.063	199.794	-.019	+.050	249.680	-.004	+.074
Feb. 16, 1918	50.016	+.018	+.027	99.752	-.004	+.040	149.800	+.016	+.079	199.813	+.019	+.069	249.708	+.028	+.102
Mar. 25, 1918	49.997	-.019	+.008	99.741	-.011	+.029	149.776	-.024	+.055	199.792	-.021	+.048	249.682	-.026	+.076
	27 to 33			27 to 34			27 to 35			27 to 36			27 to 37		
June 29, 1917	299.642			349.647			399.321			449.371			498.974		
Oct. 19, 1917	299.730	+.088	+.088	349.744	+.097	+.097	399.431	+.110	+.110	449.493	+.122	+.122	499.107	+.133	+.133
Dec. 21, 1917	299.729	-.001	+.087	349.749	+.005	+.102	399.433	+.002	+.112	449.499	+.006	+.128	499.118	+.011	+.144
Feb. 16, 1918	299.741	+.012	+.099	349.763	+.014	+.116	399.448	+.015	+.127	449.531	+.032	+.160	499.145	+.027	+.171
Mar. 25, 1918	299.725	-.016	+.083	349.744	-.019	+.097	399.435	-.013	+.114	449.517	-.014	+.146	499.139	-.006	+.165
	27 to 38			27 to 39			27 to 40			27 to 41			27 to 42		
June 29, 1917	548.913			598.460			648.691			698.831			748.405		
Oct. 19, 1917	549.043	+.130	+.130	598.571	+.111	+.111	648.774	+.083	+.083	698.894	+.063	+.063	748.479	+.074	+.074
Dec. 21, 1917	549.057	+.014	+.144	598.590	+.019	+.130	648.804	+.030	+.113	698.860	-.034	+.029	748.433	-.046	+.028
Feb. 16, 1918	549.075	+.018	+.162	598.605	+.015	+.145	648.799	-.005	+.108	698.922	+.062	+.091	748.487	+.054	+.082
Mar. 25, 1918	549.075	.000	+.162	598.602	-.003	+.142	648.789	-.010	+.098	698.889	-.033	+.058	748.462	-.025	+.057
	27 to 43														
June 29, 1917	798.169														
Oct. 19, 1917	798.263	+.094	+.094												
Dec. 21, 1917	798.206	-.057	+.037												
Feb. 16, 1918	798.283	+.077	+.114												
Mar. 25, 1918	798.266	-.017	+.097												

SURVEY RECORD, LONGITUDINAL MOVEMENT, FIG. 3.

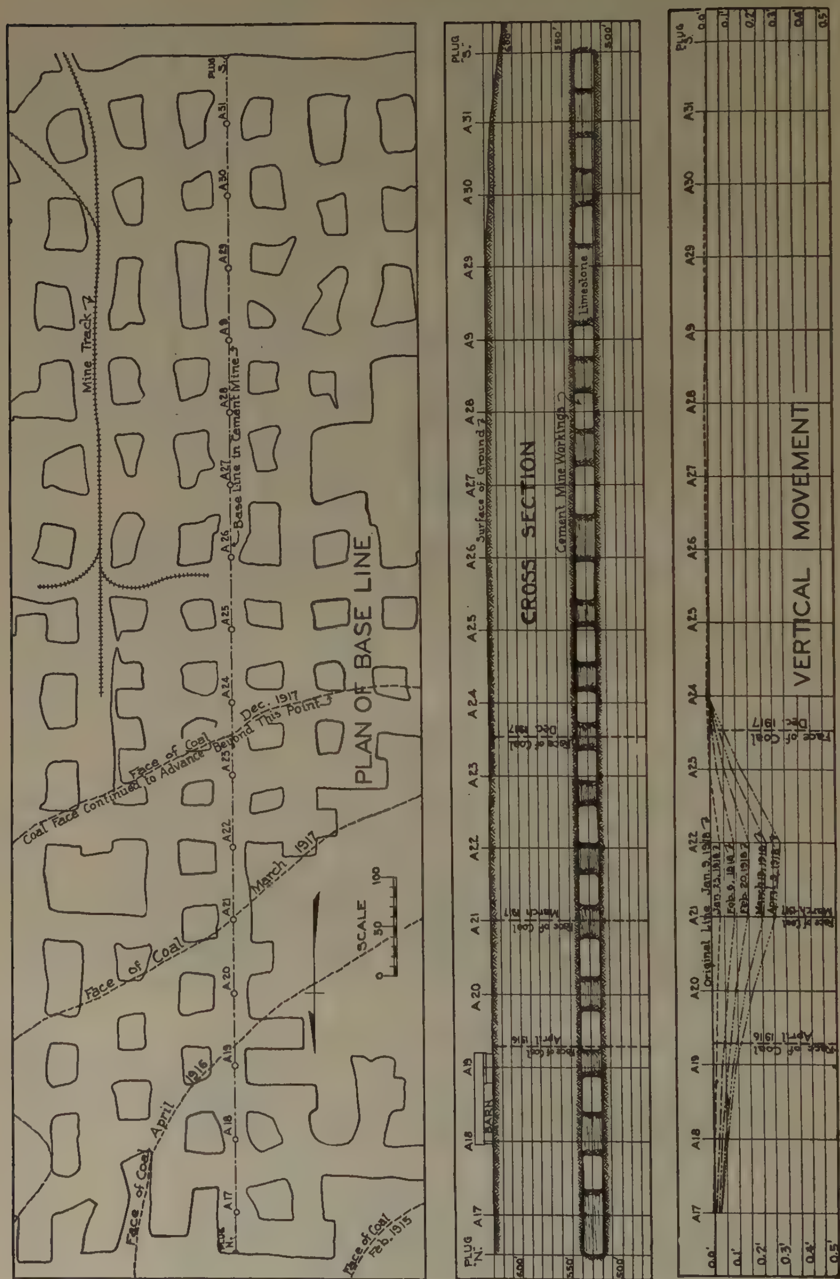

FIG. 4.—BASE LINE, STATIONS A17 TO A31, IN CEMENT WORKINGS.

Date	A-17 Elevation	A-17 Difference Last Survey	A-17 Difference Total	A-18 Elevation	A-18 Difference Last Survey	A-18 Difference Total	A-19 Elevation	A-19 Difference Last Survey	A-19 Difference Total	A-20 Elevation	A-20 Difference Last Survey	A-20 Difference Total	A-21 Elevation	A-21 Difference Last Survey	A-21 Difference Total	A-22 Elevation	A-22 Difference Last Survey	A-22 Difference Total
Jan. 9, 1918	518.003			519.076			518.272			518.773			518.280			519.513		
Jan. 23, 1918	518.008	+.005	+.005	519.068	-.008	-.008	518.260	-.012	-.012	518.755	-.018	-.018	518.245	-.035	-.035	519.479	-.034	-.034
Feb. 6, 1918	517.998	-.010	-.005	519.053	-.015	-.023	518.235	-.025	-.037	518.714	-.041	-.059	518.187	-.058	-.093	519.420	-.059	-.093
Feb. 20, 1918	517.985	-.013	-.018	519.045	-.008	-.031	518.213	-.022	-.059	518.678	-.036	-.095	518.134	-.053	-.146	[redacted]	-.058	-.151
Mar. 5, 1918	517.979	-.006	-.024	519.030	-.015	-.046	518.191	-.022	-.081	518.640	-.038	-.133	518.070	-.064	-.210	[redacted]	-.069	-.220
Apr. 3, 1918	517.980	+.001	-.023	519.025	-.005	-.051	518.181	-.010	-.091	518.610	-.030	-.163	518.019	-.051	-.261	519.232	-.061	-.281

Date	A-23 Elevation	A-23 Difference Last Survey	A-23 Difference Total	A-24 Elevation	A-24 Difference Last Survey	A-24 Difference Total	A-25 Elevation	A-25 Difference Last Survey	A-25 Difference Total	A-26 Elevation	A-26 Difference Last Survey	A-26 Difference Total	A-27 Elevation	A-27 Difference Last Survey	A-27 Difference Total	A-28 Elevation	A-28 Difference Last Survey	A-28 Difference Total
Jan. 9, 1918	518.315			519.402			519.169			519.339			517.332			517.696		
Jan. 23, 1918	518.315	.000	.000	519.413	+.011	+.011	519.174	+.005	+.005	519.346	+.007	+.007	517.340	+.008	+.008	517.706	+.010	+.010
Feb. 6, 1918	518.288	-.027	-.027	519.403	-.010	+.001	519.165	-.009	-.004	519.336	-.010	-.003	517.334	-.006	+.002	517.695	-.011	-.001
Feb. 20, 1918	518.262	-.026	-.053	519.405	+.002	+.003	519.163	-.002	-.006	519.331	-.005	-.008	517.331	-.003	-.001	517.695	.000	-.001
Mar. 5, 1918	518.227	-.035	-.088	519.403	-.002	+.001	519.163	.000	-.006	519.342	+.011	+.003	517.334	+.003	+.002	517.701	+.006	+.005
Apr. 3, 1918	518.190	-.037	-.125	519.401	-.002	-.001	519.163	.000	-.006	519.342	.000	+.003	517.334	.000	+.002	517.700	-.001	+.004

Date	A-9 Elevation	A-9 Difference Last Survey	A-9 Difference Total	A-29 Elevation	A-29 Difference Last Survey	A-29 Difference Total	A-30 Elevation	A-30 Difference Last Survey	A-30 Difference Total	A-31 Elevation	A-31 Difference Last Survey	A-31 Difference Total
Jan. 9, 1918	517.217			517.394			517.546			517.665		
Jan. 23, 1918	517.220	+.003	+.003	517.399	+.005	+.005	517.550	+.004	+.004	517.668	+.003	+.003
Feb. 6, 1918	517.222	+.002	+.005	517.400	+.001	+.006	517.548	-.002	+.002	517.673	+.005	+.008
Feb. 20, 1918	517.220	-.002	+.003	517.395	-.005	+.001	517.549	+.001	+.003	517.671	-.002	+.006
Mar. 5, 1918	517.220	.000	+.003	517.399	+.004	+.005	517.550	+.001	+.004	517.670	-.001	+.005
Apr. 3, 1918	517.221	+.001	+.004	517.397	-.002	+.003	517.548	-.002	+.002	517.666	-.004	+.001

SURVEY RECORD, VERTICAL MOVEMENT, FIG. 4.

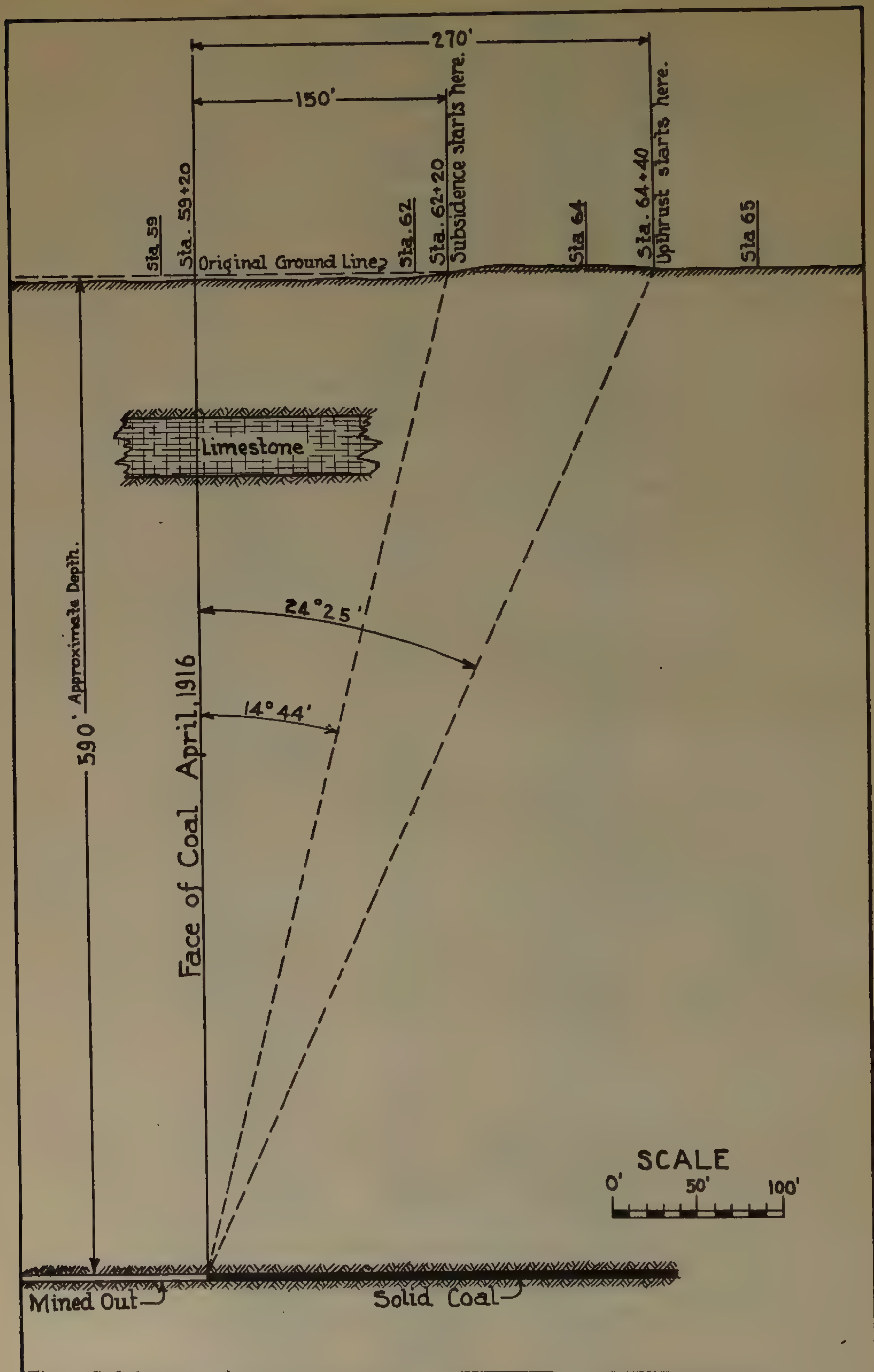

FIG. 5.—ANGLES OF DRAW IN ADVANCE OF COAL FACE. SURFACE OBSERVATIONS MADE APRIL 11, 1916. SKETCH MADE FROM DATA OF FIG. 2.

typical and is found to more or less extent in all observations covering this case.

Table 1 shows the approximate angles of draw indicated by observations on all the base lines at various time intervals.

TABLE 1.—*Approximate Angles of Draw*

Survey Date	Angle of Draw[a]	
	To Point of Subsidence	To Point of Upheaval
Fig. 2, stations 53 to 67 on surface		
Aug. 15, 1915	+56° 54′	None
Oct. 25, 1915	+54° 51′	None
Apr. 11, 1916	+14° 44′	+24° 25′
Feb. 10, 1917	+14° 16′	+30° 36′
Fig. 3, stations 27 to 43 on surface		
Aug. 19, 1915	+ 8° 12′	+34° 40′
Oct. 25, 1915	+17° 03′	+33° 52′+
Apr. 11, 1916	+15° 32′	+27° 16′+
Base line on surface, stations 17 to 25 and 124 to 131		
Oct. 24, 1917	+10° 09′	+23° 51′+
Feb. 10, 1918	+ 8° 29′	+?
Base line on surface, north T-rail to 144		
Oct. 24, 1917	+12° 31′	+27° 48′+
Feb. 10, 1918	+11° 35′	+?
Base line on surface, stations 110 to 123. Line too short to measure angle of draw.		
Base line on surface, stations 9 to 14		
Oct. 25, 1915	+11° 47′	+48° 53′+
Apr. 11, 1916	+ 7° 04′	+46° 39′
Feb. 10, 1917	+11° 02′	+?
Apr., 1917	+31° 06′	+?
July 3, 1917	+13° 26′	+?
Oct. 24, 1917	+17° 35′	+?
Jan. 5, 1918	+11° 47′	+?
Base line on surface, stations 70 to 79		
July 3, 1917	− 0° 05′	+48° 40′+
Oct. 24, 1917	− 0° 06′	+45° 00′+
Feb. 12, 1918	− 0° 35′	+?
Base line on surface approximately 45° to coal face, stations 3 to 9 and 15 to 17		
Aug. 19, 1915	+10° 17′	+41° 56′+
Oct. 25, 1915	+12° 37′	+?
Apr. 11, 1916	+ 0° 35′	+?
Feb. 10, 1917	+ 0° 17′	+?
Base line on surface, stations 95 to 109 and 145		
Jan. 11, 1918	+49° 50′	+56° 13′
Fig. 4, base line in limestone mine, stations A17 to A31		
Jan. 23, 1918	− 7° 36′	+53° 21′
Base line in limestone mine, stations A1 to A16		
Jan. 23, 1918	+27° 34′	+60° 57′
Base line in limestone mine, stations M1 to M24		
Sept. 19, 1917	+12° 32′	+?
Dec. 18, 1917	+ 0° 25′	+?
Mar. 12, 1918	− 0° 37′	+?

[a] Angles marked + are in front of the advancing longwall face, those marked − are back of the longwall face. A + after an angle indicates that the angle extended beyond the end of the base line and could not be calculated accurately. A question mark indicates that while rise in the ground was observed, the action went far beyond the base line and no angle could be calculated.

Acknowledgments

The author wishes to express his appreciation to Dr. L. E. Young, who had charge of preparing the engineering data pertaining to this case for the Marquette Cement Manufacturing Co., for allowing the use of his files in the preparation of the data presented in this paper, and also for very helpful direction in its preparation.

Dr. Young stated that much of the credit for collecting and preparing the engineering data is due Mr. R. S. Schultz, Jr., who was chief engineer of the Marquette Cement Manufacturing Co. during the period 1916 to 1918.

DISCUSSION

(George S. Rice presiding)

H. N. Eavenson, Pittsburgh, Pa., explained the present interest in the subject of Mr. Auchmuty's paper by saying that a Pittsburgh limestone company recently brought suit against a coal company, which was mining coal 35 ft. under the limestone, and approaching the limestone company's property. Interest in this case led to a review of the occurrence in Illinois, in which a large collection of data was checked over and of which Mr. Auchmuty's paper gives a small but representative proportion. The Illinois company had no objection to the publication of the data, but had been unable to find the records, which had been stored away. Mr. Eavenson said that undoubtedly more detailed work relating to subsidence has been done in this small area than anywhere else in the country. All of the surveys were checked independently by three different corps of surveyors, the figures given in the paper being averages of their results.

R. S. Schultz, Jr., East Orange, N. J., who was consulting engineer for the cement company at the time of the subsidence, and who had supervisory charge of the preparation of the engineering data, spoke of the difficulty of securing stable base lines for the observation of horizontal movement of surface monuments. After careful consideration of the problem, it was decided that the best that could be done, under the circumstances, was to use the monument that had had the maximum time for subsidence. Hence, for each line of monuments under observation, the monument that was farthest back of the coal face was used as the base. In all cases, the backsight was taken on a ball on the top of the municipal water tower in Oglesby, from ½ to 1 mile away, with a definite angle turned to the line of the monuments. The transits used for this work read to 30 sec. Standardized 200-ft. tapes, under standard tension, were used for longitudinal measurements, and these measurements were corrected for temperature and sag.

The collection of data on the horizontal movements of monuments was not started until a few months before the case was prepared for trial, therefore the number of observations was limited and the results are largely inconclusive. The data were presented for what they might be worth, without their connection with subsidence being definitely proved.

Mr. Schultz did not question the possibility that the action of frost, the drying out of the surface soil and other natural causes might have been partly responsible for the movements observed. On the other hand, the size of the monuments, the depth at which they were placed, the character of the soil, the dates of the observations

(October 19 to March 29), and particularly the nature of the movements observed, indicated the probability that relative ground movements occurred and that there was some definite connection between these movements and subsidence. This position was considerably strengthened by similar but even less extensive observations made in the mine, where questions of surface and soil conditions and temperature did not enter. Also, certain mining difficulties experienced, particularly in advance of the coal face, are extremely hard to explain except on the assumption of some such relative movement.

In all of the surveys, essential accuracy was aimed at rather than extreme precision. The three survey crews worked independently but together as far as time was concerned. Their results were carefully checked against each other and the checked results averaged for the final data.

The mine was opened in 1902 with rather irregular methods of development and a high percentage of extraction. Also, considerable quantities of the underlying shale were mined and water was allowed to accumulate in the old shale workings. Some serious difficulties had developed in the mine by 1914 but the management thought these were due to the previous methods of mining. To eliminate this possibility, it was decided to develop a large block of reserve territory to the west of the areas being worked. This development was started by driving an entry due west from the old workings into virgin limestone. This entry was driven 35 ft. wide instead of the usual 40 ft.; it opened a remarkably solid limestone and showed a perfect roof. This entry had reached a length of 760 ft. without any difficulties. On the night of Dec. 16, 1914, with the night shift working, suddenly, wtihout any warning, caving of the roof began about 100 ft. back of the face of the entry and extended so rapidly that the miners barely escaped. This caving extended both ways for the entire length of the entry and was prevented from entering the old workings only by the erection of emergency timber cogs.

This disaster convinced the management that forces outside its own control were responsible for the mining difficulties. A survey of the situation showed that the face in the longwall mine of the Oglesby Coal Co. was roughly parallel with the entry which had caved, and almost directly under it. A consultation of experts agreed that subsidence, due to longwall coal mining below, was the cause of the difficulty.

To establish the facts of subsidence, 69 monuments were established on surface and 94 underground, in January and February, 1915, under the direction of Dr. L. E. Young; 76 additional surface monuments and 148 mine monuments were established early in 1917. A considerable number of additional monuments, largely miscellaneous in character, were established in the mine in 1917 and 1918. The surface monuments were all made of concrete with a bronze pin in the top. They were 18 in. to 2 ft square and extended approximately 3 ft. below the surface, to minimize frost action The mine monuments varied from horseshoe nails in wooden plugs, set in pillars, ribs and roof, to 1-in. steel rods, 6 ft. long, grouted in the floor to concrete monuments similar to those on surface. Wherever possible, both on surface and underground, the monuments were set at regular intervals, in lines crossing the approximate position of the coal face. Every effort was made to place the various monuments so that they could develop the full facts on subsidence. The vertical position of these monuments was observed at frequent intervals, but it was not until late in 1917 that any effort was made to observe possible changes in horizontal position.

The maximum surface subsidence observed was 1.847 ft. between Feb. 23, 1915 and March 28, 1918. The maximum mine subsidence was 1.936 ft. between Jan. 21, 1915 and April 5, 1918. Probably the maximum rate of subsidence observed was 0.466 ft. in 72 days.

Mr. Schultz discussed leveling difficulties experienced in one certain part of the mine—the southwest haulage entry. In this entry, which was approximately 10

ft. wide by 7 ft. high, concrete floor monuments, with roof plugs approximately over them, had been established at 50-ft. intervals for the entire length of about 1500 ft. The direction of the entry was nearly perpendicular to the coal face. On several occasions, the surveying crews reported difficulty in securing check observations in their work in this entry, discrepancies up to 0.15 ft. being not unusual. The surveyors declared that they had observed actual vertical movement. Finally, at about 10 a. m. on Jan. 16, 1918, Mr. Schultz was called into the mine to observe this action. On looking through the instruments, at the leveling rods, he observed a vertical movement very noticeable to the eye. The movement was relatively slow and regular. It closely resembled the waving of a leveling rod, back and forth, by a rodman. No such movement of the rod occurred, however, as one rodman held the rod plumb, using a rod level, while a second man threw the light on the target.

As soon as possible after Mr. Schultz reached the entry, the three levels were set in different positions along the entry, one in advance of the approximate position of the coal face and two behind this position, and observations were made from each position. A maximum vertical movement of 0.092 ft. (1⅛ in.) between 11 a. m. and 1:51 p. m. was observed between an instrument set about 150 ft. behind the coal face and the rod held about 75 ft. in advance of the face. No movement was observed by an instrument set 300 ft. ahead of the face and this same rod. With the first instrument sighted on the rod held about 375 ft. back of the face, the maximum observed movement was 0.029 ft. (⅜ in.). The violence of the movement gradually decreased until by 4 p. m. it had ceased entirely. These observations indicated an actual vertical movement of the floor of the limestone mine, about 130 ft. below the surface.

W. O. Hotchkiss, Houghton, Mich., inquired whether the apparent movement of the floor might not have been due to refraction of the air, pointing out that the entry was the lowest part of the mine, which was kept partly closed off. He felt that there might have been stratification of the cold incoming air and the warm outgoing air, with such irregular mixing of the two currents as to produce highly refractive conditions.

Mr. Schultz said that while such refraction might have been a possible cause there were no evidences of such conditions: that the air in the entry was remarkably clear, as evidenced by sights of over 1000 ft. during the observations. The entry was used only partly for ventilation and there was no noticeable movement of the air. At this particular time, this part of the mine was not being operated and there were no movements through the entry during the time of the observations, except by the members of the survey party.

All agreed that the variability of the readings is a matter of scientific interest which should be answered if possible.

Mr. Schultz described the LaSalle limestone, where mined by the cement company, as usually a hard, crystalline limestone, 30 to 35 ft. thick, divided into four ledges by shale partings of varying thicknesses. The three lower ledges, which constitute the mined portion of the deposit, have a remarkably uniform thickness of 7 ft. each, to a total mining height of 21 ft. The parting between the "bottom" and "middle" ledges is particularly prominent, being as much as 18 in. thick in some places. The parting between the "middle" and "top" ledges is thin and not always present. The parting between the "top" ledge and the "roof" is also thin, but very persistent throughout the entire area of mining. The "roof," which numerous tests showed to have a thickness varying between 9 and 14 ft., is usually a hard, dense limestone, without partings, and makes a splendid mine roof. In areas mined in 1903 and 1904, and not disturbed by subsidence, this roof is standing today in the

same condition as when originally mined. At the extreme eastern end of the mine development, the roof changes to a so-called "hubbly" or nodular limestone. In the summers, "sweating" causes considerable "spawling" in this limited area, but this difficulty has not interfered with mining. At the time of the subsidence investigation, there had been no caving in this area, except locally and close to the outcrop. This "hubbly" roof area was far in advance of the coal face and showed none of the effects of subsidence so noticeable in other areas. Also, because of its position in relation to the remainder of the mine and to the property lines, large quantities of the underlying shale had been mined in this area; all without mining difficulty or caving. In the southwest section of the mine, about two miles from the "hubbly" roof area, the roof had changed to a "slabby" limestone with thin partings of shale. Under this roof, in areas subjected to subsidence, serious roof trouble developed even some distance ahead of the coal face. In areas not subjected to subsidence, no such troubles have developed.

The progressive effects of subsidence were studied with great care in one particular area of the mine, which had been mined for from one to four years before subsidence began. Prior to subsidence, no roof troubles had developed in this area. The mining had been done in rooms 40 ft. wide and 21 ft. high, running north and south with crosscuts between the rooms leaving roughly square pillars about 40 ft. on a side. The coal face advanced under this area diagonally. The first troubles in this area were evidenced by cracks in the black slate floor. This was followed by shelling of the round at the base of the pillars, always beginning on the corners towards the advance of the coal face and extending along the sides of the pillars from this beginning. Roof falls followed and, without exception, the roof slab broke on the sides of the rooms towards the advance of the coal face. This feature was so pronounced that all workers in the area (survey parties and inspection groups) were definitely instructed to keep to the "off" side of the rooms to avoid being hit by the falls. These falls came entirely without warning and were extremely heavy in character. In the limited area open after subsidence, the final evidence of subsidence was shelling at the base of the pillars on the sides and corners away from the advance of the coal face. These various evidences of subsidence progressed gradually across the area in a diagonal direction parallel to the advance of the face. The progressive effects of subsidence were studied with great care in this area at intervals of two to three weeks for about 18 months. When the inspections began, the area, about 6 acres, was in perfect condition, from a mining standpoint. When the inspections ended, the area was a wreck with a considerable number of caves through to surface.

In another area, an effort was made to mine the limestone after subsidence. This effort was only partly successful. The LaSalle limestone has two major systems of jointing, running roughly northeast-southwest and northwest-southeast. Under this area, the coal face had advanced almost due south. Mining of the limestone was from east to west with north and south crosscuts. The several rooms were driven 30 ft. wide and only 14 ft. high, at first. On getting into this area, it was found that both systems of jointing had been slightly opened so that the ground was extremely "blocky." Several heavy falls of the "top" ledge occurred and in several places this ledge was so drummy that it had to be taken down in actual mining. Blown-out shots were not uncommon. After a limited amount of mining, it was decided that the work was too dangerous, and all effort to mine this area was stopped. A similar condition of open joints was not found in any area in advance of the coal face.

In November, 1925, nearly 9 years after the collection of the data on subsidence, levels were run on a considerable number of mine plugs and monuments. These points showed maximum variations from −0.05 ft. to +0.24 ft., from their 1918 elevations.

H. I. Smith, Washington, D. C. (written discussion*).—The mass of the data on subsidence obtained for use in the suit between the operators of the coal mine and the operators of the limestone mine on the same land must have made difficult the task of selecting the facts presented in so brief a paper as this. I should like at some time to see all the data assembled impartially and published or, if publication is not possible, at least made available in concise manuscript form in the Engineering Societies Library. The paper represents less than 10 per cent. of the data obtained by the cement company and none of the data obtained by the coal company. The coal company admitted subsidence at the beginning of the case, and I believe the decision was based largely on the right of support.

In analyzing the surface-survey data presented in Fig. 2, and the underground-survey data in Fig. 4, I find that the two lines of monuments cross at right angles through surface station 58 and about halfway between underground stations A-17 and A-18. It is interesting to note that the average subsidence per day between stations A-17 and A-18 for the 84 days of observation underground was 0.037 ft. while the subsidence at station 58 on the surface was 0.091 ft.

The difference in subsidence on the surface and underground, and possibly in the rise of the monuments in advance of the coal face underground, may be attributed to the effect of a squeeze which was plainly observable in the floor of the cement mine on the line of the underground monuments and ahead of the advancing coal face. The squeeze also may cause some horizontal movement of the surface monuments due to the pushing down unevenly of the limestone pillars. However, some other element has entered into the case under discussion, not determined at this time. The farther the advance of the face of the longwall mine from monument 53, the greater the divergence of the radial lines. There is room for much speculation as to how much farther in advance this divergence may have continued before convergence toward their original position would have taken place. A conclusion is difficult to draw unless either the initial monument or the back site moved. Comparison of the movement in three directions—vertical, lateral and longitudinal—of stations 62 and 67 discloses no apparent relation between them, as shown below:

Vertical movement of station 62 was −0.858 ft. and of station 67 +0.002 ft. Lateral movement of station 62 was 0.50 ft. and of station 67 was 0.75 ft. Accumulated longitudinal movement from station 53 to 62 was 0.276 ft. and to 67 was 0.08 feet. These two stations were selected because of their respective positions behind and in advance of the coal face.

Again, Figs. 2 and 4 show that where the limestone had been mined under the surface monuments the lateral lines do not cross but, as indicated in Fig. 3, where no limestone was mined under the surface monuments the lines of lateral movement cross each other frequently.

A record of the caved areas and individual curves reveals that eight caves occurred at a distance from 75 to 1085 ft., averaging 637 ft. in advance of the longwall face; four caved areas ranged from 100 to 1470 ft. in advance of the coal face, averaging to near side 420 ft. and to far side 915 ft., or 668 to the center of the caved area; and four caves occurred back of face 30 to 160 ft., averaging 148.

Among those appearing as witnesses in this case on one side or the other are the following who were then members of the Institute: R. S. Schultz, Jr., L. E. Young, J. A. Garcia, H. A. Buehler, C. K. Leith, F. W. McNair, J. R. Bent, W. O. Hotchkiss, W. J. Mead, S. O. Andros, Eli T. Connor, H. I. Smith. Possibly there were others whom I do not now recall as members. A number of others took a prominent part in the case.

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* Published by permission of the Director, U. S. Geological Survey.

W. O. Hotchkiss (written discussion).—As I was employed as an expert in the case mentioned by Mr. Auchmuty, I had occasion to study the geological and mining conditions in great detail. I wish to give a little more definite impression than the paper gives of the character of the beds lying between the coal mine and the limestone mine. It might be inferred from the paper that there were rocks in this 435 to 470 ft. of thickness which were rather hard and strong. As a matter of fact, they are very weak shales, and what is called sandstone in the paper is a fine silty shale rather than a competent sandstone member. The character of much of these beds may be indicated by the fact that near by some of them are mined as fireclay by driving a drift into the hillside, wetting the bottom thoroughly and letting the clays swell up into the drift so that the heaved material can be shoveled out.

The character of these beds is important as relating to the safety of the limestone mine and also as relating to the upheaval in advance of subsidence which the paper describes. It is my opinion that there is grave doubt whether this upheaval is an actual fact or not. The amount of it is within a reasonable limit of error in the bench marks in surveys, and the probability of such an upheaval is very much lessened by the soft nature of the beds between the coal mine and the limestone mine.

The author of the paper takes it for granted that the unfavorable conditions in the limestone mine were produced by the subsidence due to the coal mining. This conclusion I cannot allow to pass without suggesting that the methods of the limestone company were at least in part to blame for the troubles at their property. They removed large areas of limestone without leaving sufficient pillars and the consequent failure of the shaly roof and of the shales underneath the limestone pillars were at least in part, I believe, responsible for their difficulties.

The actual subsidence as shown by the records of the cement company showed a maximum bending of the rock beds of 2 in. in a distance of 100 ft. According to information in standard handbooks, a bed of solid plate glass of the dimensions of the limestone bed would stand this amount of bending without cracking. Furthermore, the witnesses for the plaintiff stated that the actual horizontal "pull" due to the lengthening of the limestone bed on the curve assumed during subsidence was only 0.005 ft., or about 0.06 in. That such minute displacements and deformations of a jointed limestone formation could produce serious effects in the way of breaking the limestone or endangering safe workings seems rather hard to understand.

One important conclusion of the paper is based on what I believe to be misleading and wrongly interpreted information. This is in regard to the lateral and longitudinal movement shown in the diagrams. The fault with the survey data on which these conclusions are based is not in the accuracy of measuring distances and angles on the part of the survey, because these were carefully checked; it lies in the fact that the surveyors did not consider the difficulty of making immovable bench marks in Illinois corn-belt gumbo soil. This material will expand and contract up to as much as 20 per cent. with varying moisture conditions. Furthermore, on the hillsides where some of the monuments were located the monuments were subjected to the natural hillside creep with which every geologist is familiar. On Fig. 3, monument 43 was on a hillside slope and almost continuously throughout the series of readings moved down the hill.

Mr. Schultz has stated that the back sight for measuring the angles on which the lateral movement was based was the top of a steel water tower some distance away. The most probable interpretation of these lateral movements is that this water tower behaved in a well-known fashion and changed the relative lengths of its legs and therefore the position of its top with varying temperature conditions.

To me, as a geologist, it seems preposterous to assume that rocks of this or any other character wholly supported on all sides except from underneath, subjected to a gradual drop of an inch or so per month for a total subsidence of approximately

18 in., could take on a lateral movement of as much as 9 in. in two months. Data which are as open to criticism as are these are not sufficient evidence on which to draw any reliable conclusion that such lateral movement actually did occur. It is possible, of course, that lateral movement takes place, but if it did in the instances cited in the paper, it was of much less magnitude than indicated and the data are not sufficiently accurate to prove its existence.

The writer of the paper is not to be criticized for these diagrams or the interpretation. He simply accepted at face value the information furnished and did not have the opportunity to study the conditions on the ground or to know conditions that would have enabled him to criticize the data more intelligently.

My purpose in going into this discussion is to prevent the assumed facts of lateral movement in these diagrams from being accepted at face value without at least having an alternative interpretation of the data presented.

R. L. Auchmuty (written discussion).—I had no personal connection with the case. A large amount of data concerning the case became available to the Subcommittee on Bituminous Coal Mining, and I was asked to prepare a few typical illustrations to present in a paper, the idea being merely to present the engineering data, without making any personal interpretation of the recorded observations. Portions of the paper are quotations from the decision handed down in the case and I tried to keep my own opinions out of the text. No "conclusions" were expressed.

Mr. Schultz seems to have covered thoroughly all debatable points brought up by Dr. Hotchkiss. In view of the great number of observations made and the apparent care with which they were made, I would rather accept the recorded observations of ground movement than any personal theory or opinion of how they should take place. I cannot conceive of subsidence without some lateral and longitudinal movement. This is easily observed in the opening of large surface cracks over subsiding areas. I cannot vouch for the accuracy of observations of horizontal movements shown in the paper, but even though they were 25 per cent. wrong there is still sufficient evidence that these movements occurred. Recently some observations have been received which, under entirely different geological conditions from those of Illinois, show indications that horizontal movements have taken place.

Effect on Buildings of Ground Movement and Subsidence Caused by Longwall Mining

By Wallace Thorneycroft, Whimple, Devon, England

(New York Meeting, February, 1931)

Foreword

This paper by Mr. Thorneycroft, Past President of the Institution of Mining Engineers (Great Britain), and chairman of its Subsidence Committee, is a valuable contribution to the assemblage of data on ground movement and subsidence from coal mining. It deals with the effects of advancing longwall, a method almost universal in Europe but as yet little used in the United States outside of Northern Illinois and in a few scattered places elsewhere in thin coal beds, as in the Lexington field, Missouri, Cañon City field, Colorado, and the Birmingham field, Alabama.

Although by advancing longwall, a maximum recovery of coal is obtained and it has to be employed where there are weak shale roofs, especially in deep mines, it is a more expensive method of operation unless the mining conditions are suitable, which is unusual in the United States. On the other hand, there is a growing use in this country of what is essentially retreating longwall, with an approximately straight break-line as described by Rayburn.[1] With this method, in which the pillars are substantial, it was shown that the ground movement effect is practically identical with that of advancing longwall, hence the value to mining engineers of the precise data given by Mr. Thorneycroft.

His study is unusual in several respects; it deals with conditions connected with the mining of a coal bed 20 in. thick, about 490 ft. below the surface, over a mined-out area in a 5-ft. bed about 90 ft. below, which had been mined about 12 years earlier.

Both mines passed under Plean House, a building of massive cut stone construction built over a century ago, in which Mr. Thorneycroft resided throughout the under-mining period. Slight cracks opened in the walls as the longwall face approached the house in mining the upper seam, and closed after the face passed and subsidence took place. In the northwest wall, the last wall the face passed under, some stone blocks were broken, which was attributed to a strike period when the advance of face came nearly to a standstill. Other than this there was no perceptible permanent damage, as indicated by the photographs taken by the undersigned in 1923 (Figs. 1 and 2).

Mr. Thorneycroft used a novel means of studying the successive positions of the corners of the building: he hung plumb bobs from the eaves and traced the paths of the respective plumb bob points on level surfaces. The angles made by the house walls with the plumb bob lines gave measures of the slope of the subsidence wave at the respective points.

A slight upward wave was indicated by the surveys ahead of the subsidence, which would appear to be due to a tilting over of the massive sandstone stratum in the roof measures near the surface, the edge of the coal face acting as the fulcrum.

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[1] J. M. Rayburn: Subsidence in Thick Freeport Coal. *Trans.* A. I. M. E., Coal Div. (1930) 144–148.

The Ground Movement and Subsidence Committee is fortunate in that Mr. Thorneycroft took so much pains, at the request of its chairman, to prepare this valuable paper.

George S. Rice, *Chairman*,
A. I. M. E. Ground Movement and Subsidence Committee.

THE first phase of the problem of subsidence discussed in this paper is the effect on the surface of working coal seams by the ordinary longwall method of complete extraction in an area free from large faults.

The first observation is the fact that in advance of the face, and before subsidence begins, there is a slight rise of the surface. This was first observed by me about 25 years ago when taking out a 5-ft. seam of coal below my own house (Plean House) and in nearly every case that I have had surveyed since, I have found the same thing occur.

FIG. 1. FIG. 2.

FIG. 1.—PLEAN HOUSE, STIRLINGSHIRE, SCOTLAND, SOUTHWEST END.

This end was considerably cracked and stones were replaced as shown by lighter color. The management of Plean colliery attributed the greater cracking at this end to slow advance of face during a seven-week strike in 1920.

FIG. 2.—PLEAN HOUSE, SOUTHEAST FRONT AND NORTHEAST END.

Longwall faces of Plean colliery approached front parallel with it. Photograph made after subsidence was practically complete.

Photographs by George S. Rice.

The apparent leaning of the walls is due to foreshortening in the photographs.

I attribute this rise to the "beam" idea stated by R. D. Hall,[2] and it is well illustrated by taking an ordinary ivory scale about 12-in. long, laying it on a table with about 9 in. overhang and firmly holding the end on the table while depressing the overhanging part. The edge of the table represents the coal face and a slight rise from the table will be observed beyond the edge. Fig. 3, which is not to scale but typical of the observation, indicates a longwall working of a seam, say 60 in.

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[2] R. D. Hall: Discussion of paper by H. W. Montz and R. V. Norris: Subsidence from Anthracite Mining. *Trans.* A. I. M. E., Coal Div. (1930) 122.

thick, which has passed under a house (Figs. 1 and 2). The section, Fig. 5, shows the seam with other coal seams, beds of sandstone and shale above it with the usual layer of boulder clay on the surface.

The first movement of the surface resulting from mining (Fig. 3) is generally a slight rise, which is observed at *A*. The angle of draw,

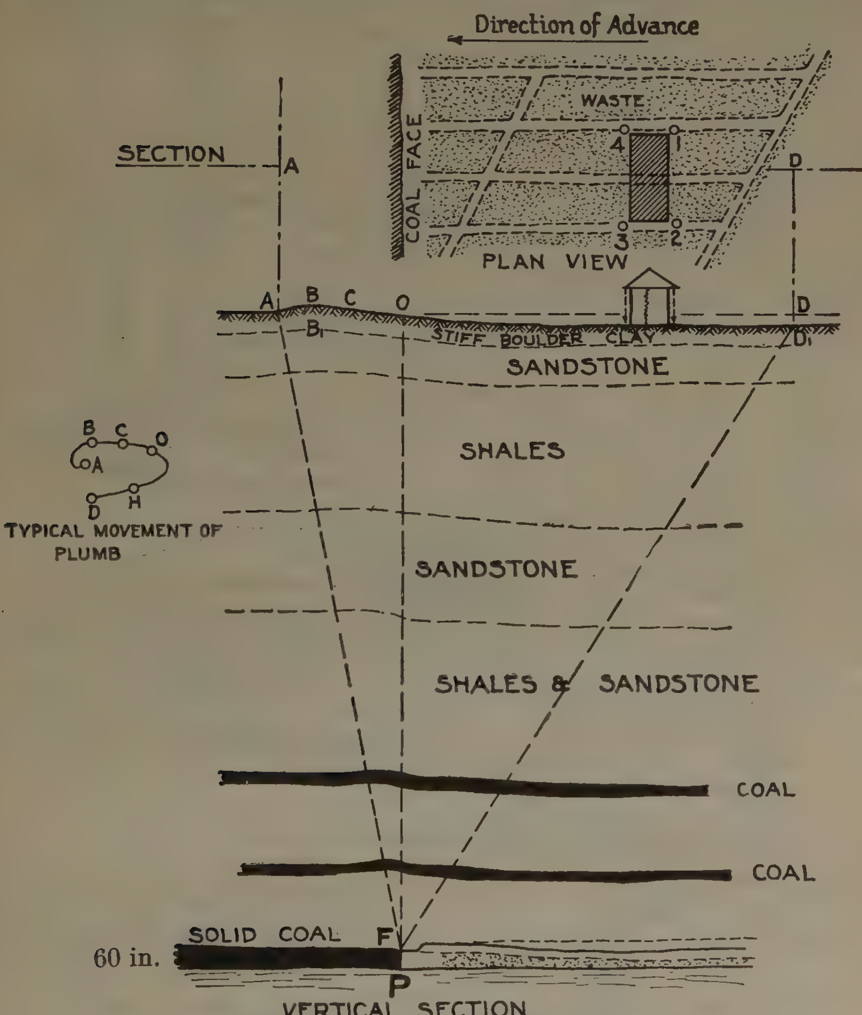


Fig. 3.—Diagrammatic study of mechanics of subsidence.

AFO, varies with many conditions; *e. g.*, depth, dip of strata, speed of advance of face, character of the rocks and of the surface deposit; 12° to 15° is common. The maximum rise at *B* that I have observed is about 2 in. when extracting a seam 5 ft. thick. Subsidence generally begins at *C* in front of the face. The distance from *O* to *A* is often referred to as the

"draw." The maximum subsidence is at D and the time before subsidence is complete at D varies with depth, et cetera.

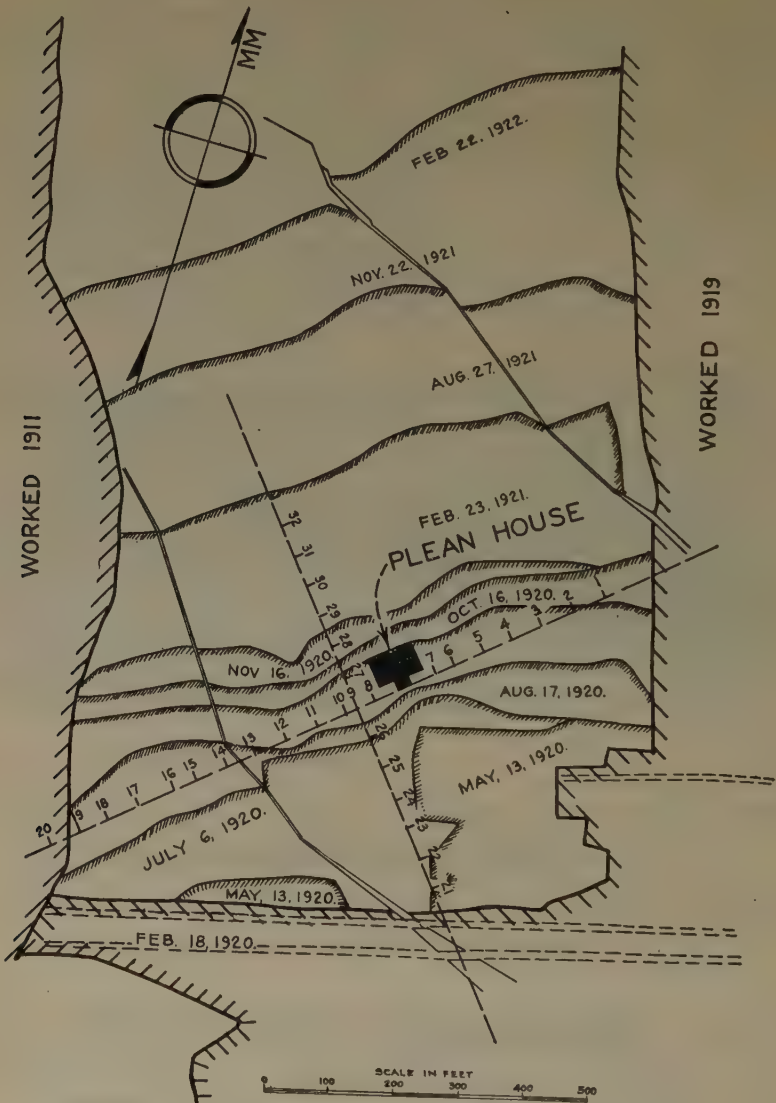


FIG. 4.—PLAN OF 20-INCH WORKINGS.

Cracks will often begin to open in the walls of a house when the face in relation to the house is as B is to F and the curve of subsidence is convex and will begin to close some months before subsidence is complete when the curve of subsidence reverses.

For study of the effect of ground movement, I commend the following method of observing the movements of a building in addition to leveling: Hang plumb lines at the four corners of the building, as indicated in Fig. 10, and plot the movements of the respective plumb bobs. Fig. 3 shows a typical result. You will observe that between A and B the house will be tilted away from the face by the slight rise and, apart from side movements, which are usual, it will return to the plumb about

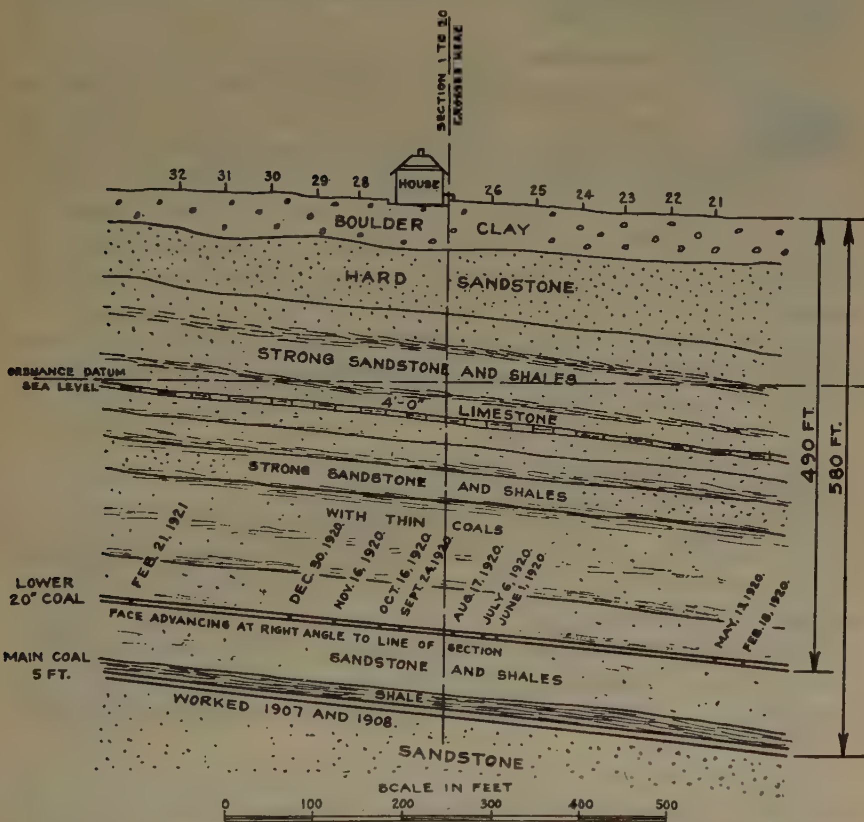


FIG. 5.—SECTION OF STRATA UNDER PLEAN HOUSE.

B, tilting backward thereafter until the curve of the wave of subsidence reverses.

The walls of the building at right angles to the line of the face are in tension in the tilting forward stage, but when the tilting is reversed the cracks begin to close and the walls may be considered as under compression, and the house returns to plumb more or less when the face in relation to the house is as F is to D. The maximum subsidence at D is about six-tenths of the thickness of coal seam, but this of course varies. Much can be learned by watching the cracks and movements of the four plumb lines on any large building.

Working of Second Seam

When a second seam is worked in the same area, the results are somewhat different and often more severe on the house.

The effect of working a lower seam first by the longwall method on the seams above is small as affecting their workability, but the coal is usually harder to get by hand and cutting machines may be required for holing. The roof conditions of the upper seam are sometimes improved. This proves that the strata observed bend but do not break unless the face of the lower seam has stopped for a considerable time. In that case, the breaks above the edge of the solid coal in the lower seam are generally observed in the upper seam and sometimes at the surface.

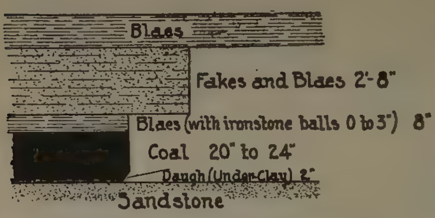


Fig. 6.—Section of lower 20-inch coal seam.

The effect of working the upper seam on the main roads in the previously worked seam below is considerable and necessitates heavy repairs in the lower seam as a rule.

Detailed Observations of Subsidence at Plean, 1920–22

Referring to the diagrams of actual observations, Figs. 4 to 13 refer to the same Plean area as Fig. 3, but apply to the subsequent working of the upper seam called the Lower Twenty Inch Coal. Most of the original observations referred to were made while working the main coal in this area, but detail records of these and some observations in other areas have been mislaid. Fig. 4 is a plan of the workings of the lower 20-in. seam; Fig. 5, the section of the strata and Fig. 6, the detail section of the seam.

In working this seam the face was not opened up on the desired line until about the first of June, 1920, and some subsidence due to the workings prior to May, 1920 had caused the north end (Fig. 10) of the house to move before observations began. Also, the strike which took place from October 16 to November 8 caused irregularity of movement. This is indicated by the small advance in the longwall face (Fig. 4) and by subsidence curves in Fig. 7.

In Table 1 are shown details of the leveling observations at the points along two lines of section indicated on Fig. 4. These lines are at right angles to one another and intersect at point 9 near the Plean House. Line 21 to 32 is at right angles to the successive positions of the longwall face and 1 to 20 parallel with them. Fig. 7 indicates the curves of subsidence. It is a profile of line 21 to 32 and shows the levels plotted

with exaggerated vertical scale at each of the dates taken and the position of the face at each date. The method of plotting is to take the horizontal initial line as the level of each point before any subsidence

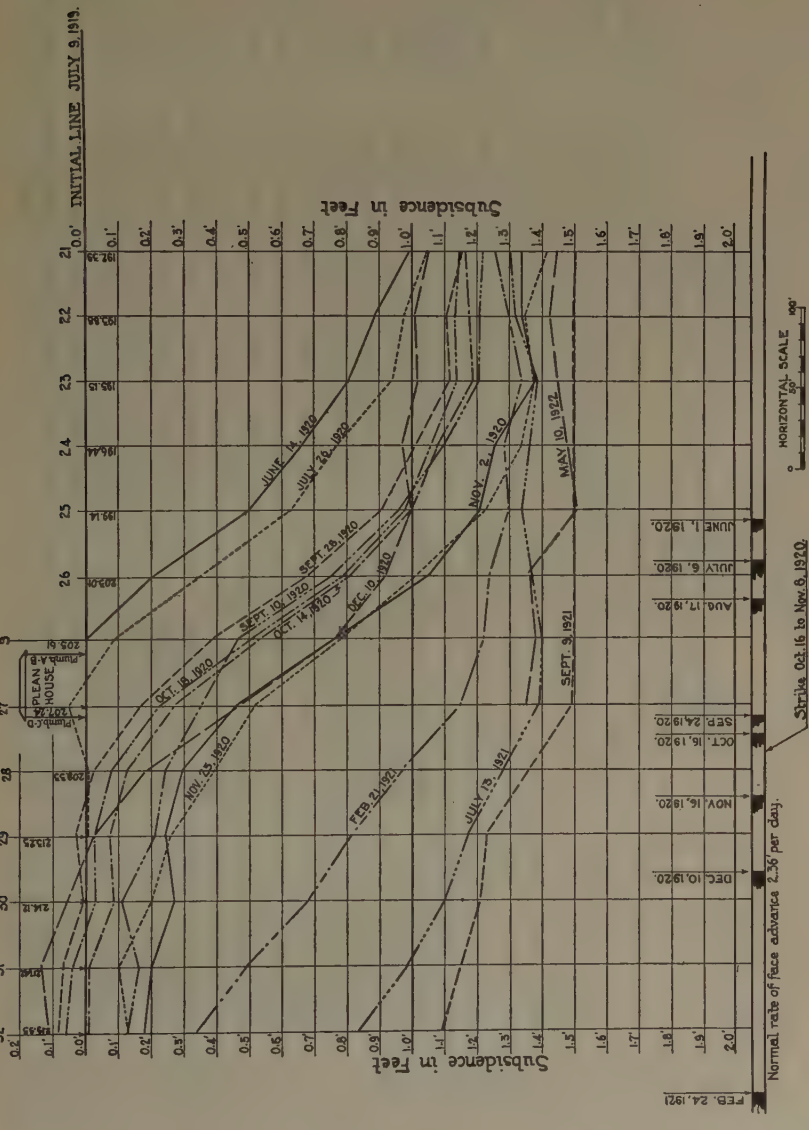


FIG. 7.—MOVEMENT ALONG LINE 21 TO 32.

began. On this section the initial levels were taken about June, 1919, but not again until June, 1920. A slight rise is shown in some cases, but the evidence of rise is not so conclusive as on the mislaid plans.

TABLE 1.—*Elevation of Points on Survey Lines at Plean*

						STRIKE PERIOD Oct. 16, 1920 to Nov. 8, 1920					STRIKE PERIOD April 1, 1921 to July 7, 1922				DIFFERENCE June 14, 1920 to Sept. 9, 1921	
		June 14 1920	July 26 1920	Sept. 10 1920	Sept. 28 1920	Oct. 14 1920	Oct. 18 1920	Nov. 2 1920	Nov. 25 1920	Dec. 10 1920	Feb. 21 1921	July 13 1921	Sept. 9 1921	May 10 1922		
EAST & WEST	Peg 1	232.05	231.98	231.87	231.84	231.81	231.80	231.62	231.68	231.71	231.17	230.94	230.85	231.14	1.20	
	" 2	231.10	231.00	230.89	230.82	230.81	230.79	230.62	230.70	230.72	230.14	229.92	229.82	230.14	1.28	
	" 3	222.69	222.62	222.48	222.48	222.35	222.33	222.14	222.21	222.22	221.66	221.46	221.38	221.65	1.31	
	" 4	214.35	214.35	214.10	214.08	213.94	213.91	213.75	213.77	213.78	213.25	213.05	212.99	213.22	1.36	
	" 5	208.74	208.73	208.49	208.46	208.33	208.30	208.14	208.11	208.12	207.65	207.46	207.39	207.44	1.35	
	" 6	208.18	208.13	207.90	207.86	207.72	207.69	207.51	207.48	207.51	206.99	206.95	206.89	206.40	1.29	
	" 7	207.81	207.76	207.46	207.49	207.32	207.32	207.12	207.09	207.11	206.68	206.49	206.42	206.58	1.39	
	" 8	207.43	207.37	206.99	207.07	206.96	206.96	206.70	206.68	206.71	206.25	206.04	205.96	206.10	1.47	
	" 9	205.61	205.53	205.15	205.23	205.08	205.11	204.85	204.83	204.85	204.39	204.21	204.11	204.23	1.50	
	" 10	203.73	203.67	203.26	203.34	203.19	203.22	202.95	202.93	202.94	202.51	202.35	202.26	202.35	1.47	
	" 11	200.45	200.33	199.95	200.01	199.85	199.89	199.60	199.52	199.54	199.19	199.07	198.95	198.96	1.50	
	" 12	197.38	197.25	196.79	196.84	196.68	196.71	196.41	196.28	196.29	196.10	195.63	195.86	195.93	1.52	
	" 13	192.13	191.86	191.46	191.45	191.27	191.30	191.00	190.89	190.93	190.78	190.30	190.51	190.56	1.62	
	" 14	184.76	184.59	184.07	184.07	183.90	183.93	183.63	183.49	183.56	183.51	183.08	183.29	183.33	1.47	
	" 15	179.42	179.20	178.69	178.68	178.46	178.57	178.27	178.11	178.19	178.22	177.74	177.95	177.87	1.47	
	" 16	174.36	174.19	173.75	173.75	173.57	173.68	173.41	173.19	173.29	173.38	172.93				
	" 17	183.36	183.21	182.74	182.74	182.62	182.70	182.41	182.15	182.26	182.43	182.01	182.20	182.14	1.16	Influence of
	" 18	186.21	186.10	185.78	185.85	185.72	185.79	185.53	185.32	185.45	185.56	185.16	185.30	185.29	.91	Old Ribside
	" 19	188.05	187.87	187.76	187.64	187.71	187.82	187.53	187.40	187.55	187.56	187.21	187.36	187.41	.69	Previous Subsidence
	" 20	191.05	190.99	190.88	190.98	190.85	190.97	190.68	190.62	190.66	190.73	190.38	190.52	190.65	.47	

NORTH & SOUTH	" 21	191.59	191.56	191.17	191.23	191.22	191.24	191.06	190.97	191.34	191.13	191.08	190.88	190.94	.51	
	" 22	192.99	192.90	192.67	192.77	192.70	192.75	192.54	192.53	192.87	192.58	192.56	192.38	192.45	.61	Subsidence Had
	" 23	194.34	194.21	193.94	194.03	193.96	194.01	193.77	193.77	194.13	193.81	193.76	193.65	193.70	.69	Begun Before
	" 24	195.78	195.67	195.54	195.43	195.36	195.30	195.18	195.10	195.47	195.15					Levels Were Taken
	" 25	198.64	198.51	198.18	198.23	198.14	198.16	197.93	197.71	198.14	197.84	197.80	197.64	197.63	1.00	
	" 26	202.81	202.67	202.36	202.33	202.20	202.22	201.96	202.00	202.09	201.77	201.64	201.61	201.65	1.30	
	" 27	207.24	207.29	206.89	207.08	206.98	207.02	206.78	206.73	206.77	206.09	205.85	205.75	205.89	1.49	
	" 28	209.53	209.32	209.09	209.51	209.21	209.26	209.04	208.94	209.16	208.36					
	" 29	213.25	213.24	213.04	213.28	213.18	213.23	213.01	212.99	213.23	212.44	212.08	212.02		1.28	
	" 30	214.12	214.81	214.01	214.15	214.04	214.09	213.85	213.92	214.17	213.44	213.02	212.91		1.21	Subsidence
	" 31	217.42	218.12	217.26	217.49	217.41	217.46	217.22	217.32	217.56	216.93	216.43				Not Complete
	" 32	219.95	220.65	219.82	220.03	219.94	220.01	219.77	219.82	220.06	219.61	219.11	218.86	218.91	1.01	
Front Left Hand Corner		200.91	200.83	200.47	200.57	200.41	200.45	200.19	200.14	200.16	199.67	199.47	199.39	199.49	1.52	
Back Left Hand Corner		201.25	201.19	200.90	201.08	200.97	201.00	200.77	200.70	200.72	200.02	199.77	199.69	199.43	1.52	
Front Door Step		207.87	207.81	207.44	207.54	207.40	207.41	207.19	207.10	207.15	206.74	206.55	206.44	206.62	1.25	Squeezed up by Thrust
Front Right Hand Corner		200.70	200.63	200.56	200.36	200.19	200.20	199.97	199.94	199.97	199.52	199.39	199.25	199.38	1.45	
Back Right Hand Corner		200.94	200.94	200.73	200.74	200.62	200.61	200.43	200.38	200.42	199.79	199.55	199.45	199.55	1.49	

Some of the anomalies may be due to mistakes of the surveyor, but most of the irregularity is due to the strike of October, 1920.

Figs. 8 and 9 show details of the movements of the plumbs hanging on wires 39 ft. long. at the dates stated. Obviously the movement had begun when the first detailed observation was taken on the first of June.

Fig. 11 shows an elevation of the house and the position of the plumb lines in relation to points on line parallel with faces (Fig. 4) and point 27 on line at right angles to faces. The vertical projections show the swing of the top of the house to northwest and to southeast. These are plotted on the assumption that the position of the walls when movement began was approximately the same as when subsidence ceased. The plumb lines indicate a swing to the northeast also between October and January.

Fig. 11 shows the vertical movements of points on the line 6 to 9 in front of the house similar to Fig. 7. Fig. 12 shows the vertical movements of a point on the front doorstep in relation to time and the position of the face. Fig. 13 similarly shows the movements of point 27.

The irregularities due to the strike are clearly shown on these two curves; the movement of the plumb lines was checked. Because of these irregularities, the stoppage of the face caused more damage than would otherwise have been done.

Between the twenty-seventh of November and the first of January compression was indicated by the breaking and lifting of the paving stones and the movement of the plumbs showed the reversal of the curve of subsidence.

Effect of Mining at Garthamlock Colliery

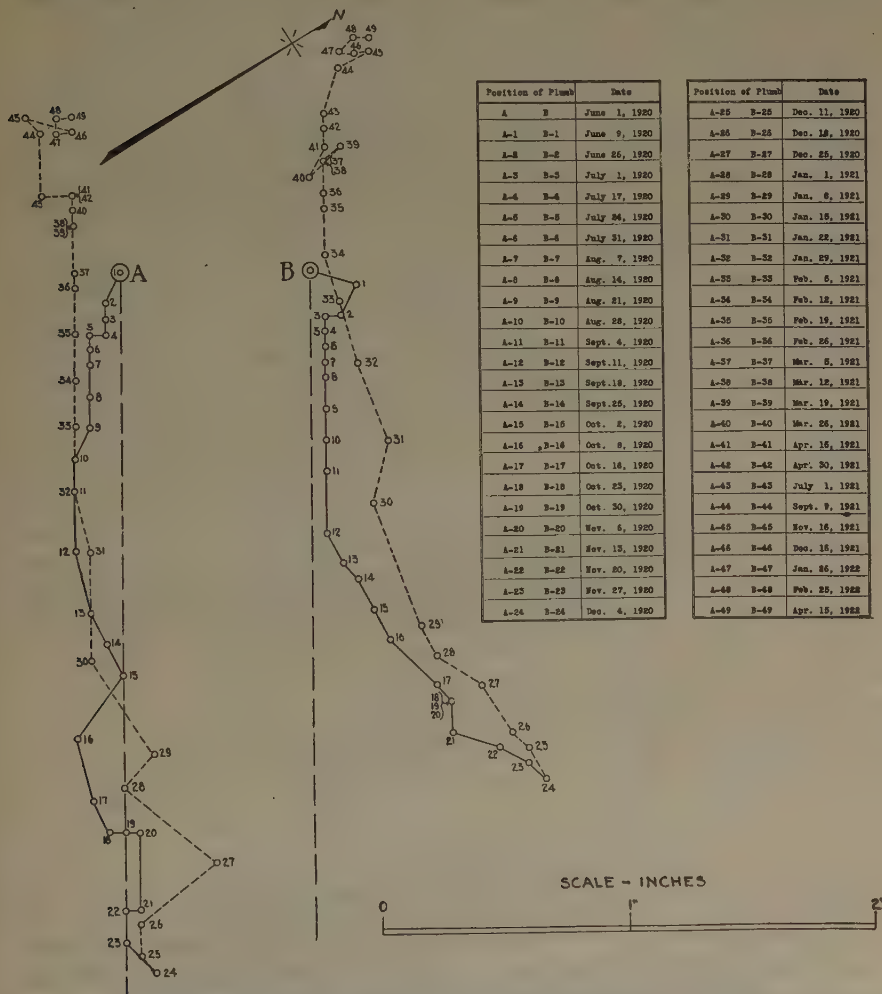
The method adopted in 1913 to observe the effect of the working of a thin seam below some massive water towers is shown in Figs. 14, 15 and 17 to 19. The irregularities in mine layout are due to a volcanic intrusion, a "whin" or dike through the coal seam in the Kiltongue workings.

The thickness of the seam was about 2 ft.; its section is shown in detail on Fig. 14. The depth of the seam from the foundation of the towers was about 68 fathoms (408 ft.). The section of the strata is shown in Fig. 15. The progress of the working faces approaching the towers is shown on Fig. 16.

The curve of subsidence, Fig. 17, is plotted by months on the horizontal scale and the measured subsidence of point *B* on the vertical scale. The position of point *B* is shown in Figs. 16 and 18. The relative positions of the advancing coal face are shown along two lines. One line is at right angles to the line of face on bearing N. 73° W., the other is on bearing N. 60° E., which is marked on the plan view, Fig. 16, and shows the widening of the excavation to the northeast as the face advances. Both lines pass through the plumb line *B* and the face

was vertically below *B* about November, 1913, and this point is used to correlate the face with the curve of subsidence of the point *B*.

The angle made by the plumb line from the original position is shown, Fig. 17, at various dates and the actual movements of the plumb line



Position of Plumb		Date
A	B	June 1, 1920
A-1	B-1	June 9, 1920
A-2	B-2	June 26, 1920
A-3	B-3	July 1, 1920
A-4	B-4	July 17, 1920
A-5	B-5	July 24, 1920
A-6	B-6	July 31, 1920
A-7	B-7	Aug. 7, 1920
A-8	B-8	Aug. 14, 1920
A-9	B-9	Aug. 21, 1920
A-10	B-10	Aug. 28, 1920
A-11	B-11	Sept. 4, 1920
A-12	B-12	Sept. 11, 1920
A-13	B-13	Sept. 18, 1920
A-14	B-14	Sept. 25, 1920
A-15	B-15	Oct. 2, 1920
A-16	B-16	Oct. 8, 1920
A-17	B-17	Oct. 16, 1920
A-18	B-18	Oct. 23, 1920
A-19	B-19	Oct. 30, 1920
A-20	B-20	Nov. 6, 1920
A-21	B-21	Nov. 13, 1920
A-22	B-22	Nov. 20, 1920
A-23	B-23	Nov. 27, 1920
A-24	B-24	Dec. 4, 1920

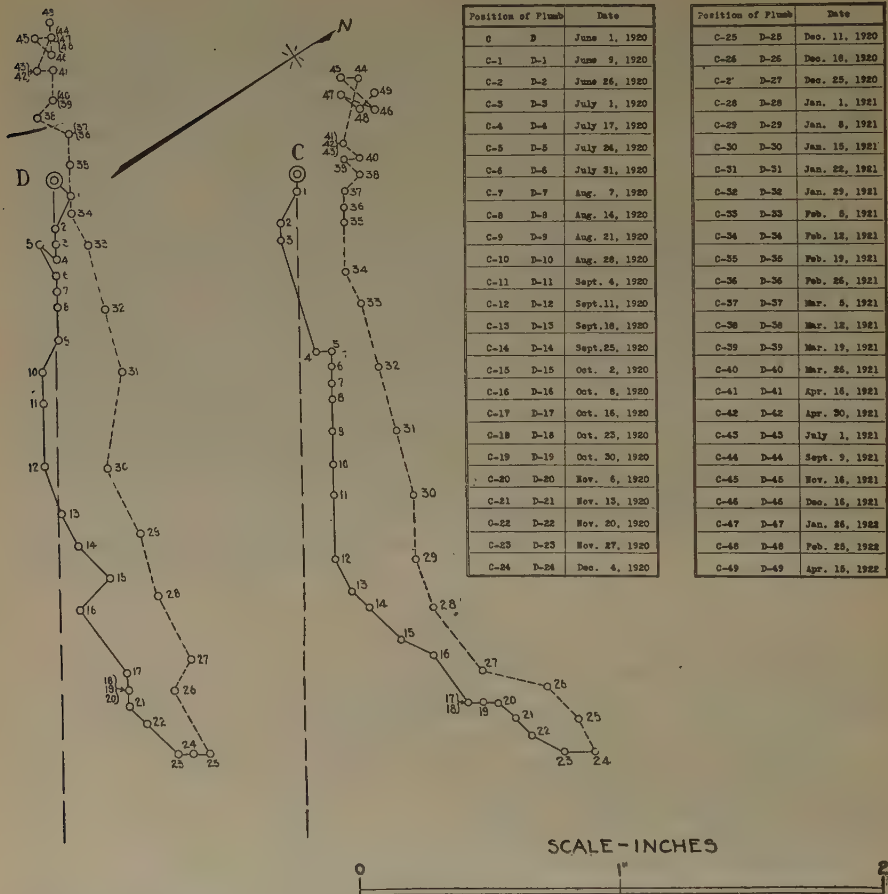
Position of Plumb		Date
A-25	B-25	Dec. 11, 1920
A-26	B-26	Dec. 18, 1920
A-27	B-27	Dec. 25, 1920
A-28	B-28	Jan. 1, 1921
A-29	B-29	Jan. 8, 1921
A-30	B-30	Jan. 15, 1921
A-31	B-31	Jan. 22, 1921
A-32	B-32	Jan. 29, 1921
A-33	B-33	Feb. 5, 1921
A-34	B-34	Feb. 12, 1921
A-35	B-35	Feb. 19, 1921
A-36	B-36	Feb. 26, 1921
A-37	B-37	Mar. 5, 1921
A-38	B-38	Mar. 12, 1921
A-39	B-39	Mar. 19, 1921
A-40	B-40	Mar. 26, 1921
A-41	B-41	Apr. 16, 1921
A-42	B-42	Apr. 30, 1921
A-43	B-43	July 1, 1921
A-44	B-44	Sept. 9, 1921
A-45	B-45	Nov. 16, 1921
A-46	B-46	Dec. 16, 1921
A-47	B-47	Jan. 26, 1922
A-48	B-48	Feb. 25, 1922
A-49	B-49	Apr. 15, 1922

FIG. 8.—MOVEMENT OF PLUMB BOBS A AND B WITH REFERENCE TO BOTTOM OF RESPECTIVE WALLS OF HOUSE. LENGTH OF PLUMB LINES, 39 FEET.

at the bottom are plotted full size on the plan from the table of observations given in detail, Fig. 19.

The first observations were taken in April, 1913, of plumbs and in May, 1913, for levels. At this date the face was only 70 ft. or thereby from *B* on plan, and there may have been a slight movement before that date as the levels and plumbs show no distinct rise.

It will be observed that there was very little movement of the plumb line until after June 16, 1913, but on Aug. 12, 1913, the plumb showed a movement of the point of suspension of 3¼ in. S. 76° E. equivalent to an angle of 0° 23′. (See Table 2.) The level of the base at *B* oscillated slightly between June and August and thereafter started to fall irregularly



Position of Plumb		Date
C	D	June 1, 1920
C-1	D-1	June 9, 1920
C-2	D-2	June 26, 1920
C-3	D-3	July 1, 1920
C-4	D-4	July 17, 1920
C-5	D-5	July 24, 1920
C-6	D-6	July 31, 1920
C-7	D-7	Aug. 7, 1920
C-8	D-8	Aug. 14, 1920
C-9	D-9	Aug. 21, 1920
C-10	D-10	Aug. 28, 1920
C-11	D-11	Sept. 4, 1920
C-12	D-12	Sept.11, 1920
C-13	D-13	Sept.18, 1920
C-14	D-14	Sept.25, 1920
C-15	D-15	Oct. 2, 1920
C-16	D-16	Oct. 8, 1920
C-17	D-17	Oct. 16, 1920
C-18	D-18	Oct. 23, 1920
C-19	D-19	Oct. 30, 1920
C-20	D-20	Nov. 6, 1920
C-21	D-21	Nov. 13, 1920
C-22	D-22	Nov. 20, 1920
C-23	D-23	Nov. 27, 1920
C-24	D-24	Dec. 4, 1920

Position of Plumb		Date
C-25	D-25	Dec. 11, 1920
C-26	D-26	Dec. 18, 1920
C-2'	D-27	Dec. 25, 1920
C-28	D-28	Jan. 1, 1921
C-29	D-29	Jan. 8, 1921
C-30	D-30	Jan. 15, 1921
C-31	D-31	Jan. 22, 1921
C-32	D-32	Jan. 29, 1921
C-33	D-33	Feb. 5, 1921
C-34	D-34	Feb. 12, 1921
C-35	D-35	Feb. 19, 1921
C-36	D-36	Feb. 26, 1921
C-37	D-37	Mar. 5, 1921
C-38	D-38	Mar. 12, 1921
C-39	D-39	Mar. 19, 1921
C-40	D-40	Mar. 26, 1921
C-41	D-41	Apr. 16, 1921
C-42	D-42	Apr. 30, 1921
C-43	D-43	July 1, 1921
C-44	D-44	Sept. 9, 1921
C-45	D-45	Nov. 16, 1921
C-46	D-46	Dec. 16, 1921
C-47	D-47	Jan. 26, 1922
C-48	D-48	Feb. 25, 1922
C-49	D-49	Apr. 15, 1922

FIG. 9.—MOVEMENT OF PLUMB BOBS C AND D WITH REFERENCE TO BOTTOM OF RESPECTIVE WALLS OF HOUSE. LENGTH OF PLUMB LINE, 39 FEET.

during the period of observation to Oct. 27, 1914, when war prevented further time being spent on this work.

The maximum angle of the plumb observed was on May 15, 1914, when the face was 170 ft. past the point *B* and the curve of subsidence began to flatten, and if cracks had been made they would have begun to close. The last observation in June of the plumb indicated the commencement of the return towards the original vertical position of the tower.

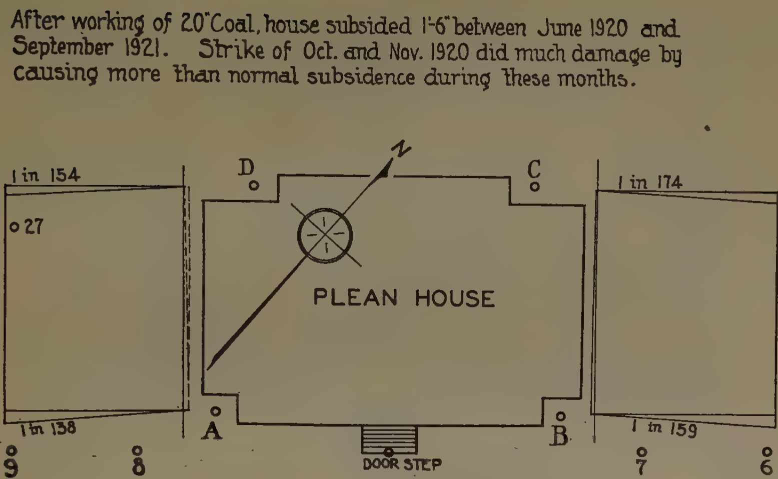


FIG. 10.—PLAN AND ELEVATIONS SHOWING POSITION OF PLUMBS IN RELATION TO POINTS 6 TO 9 AND 27 TO 9; ALSO SWING OF HOUSE AS INDICATED BY PLUMBS.

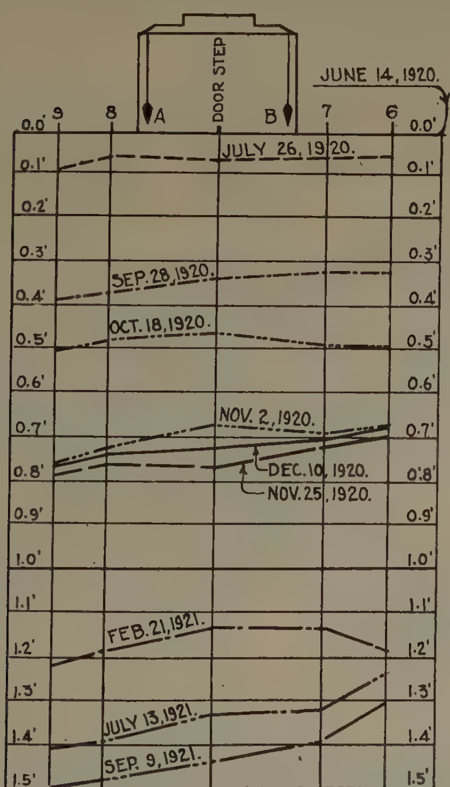


FIG. 11.—MOVEMENT ALONG LINE 6 TO 9.

The foundation of the tower was annular and not a rigid mass (Fig. 20) but the rock at the surface was solid whinstone (eruptive rock) and the underground workings were intersected by the volcanic intrusion

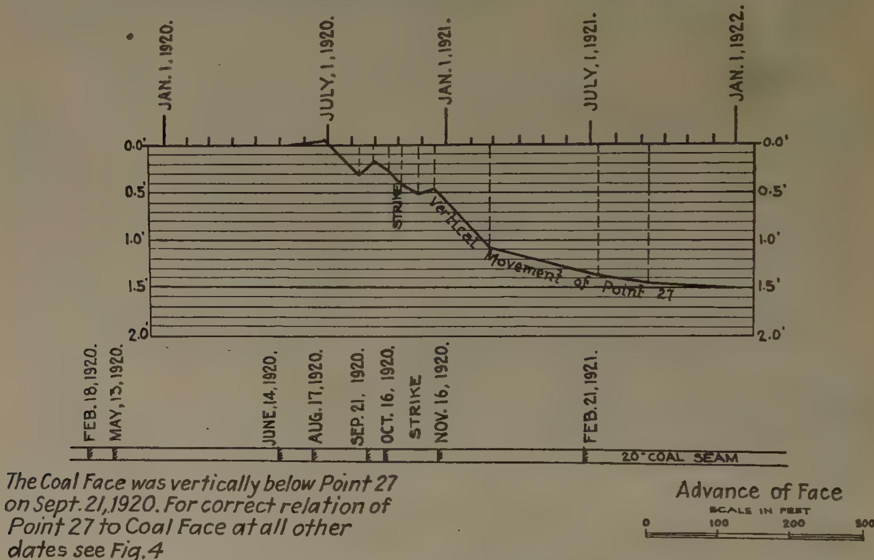


FIG. 12.—MOVEMENT OF DOORSTEP IN RELATION TO TIME AND COAL FACE.

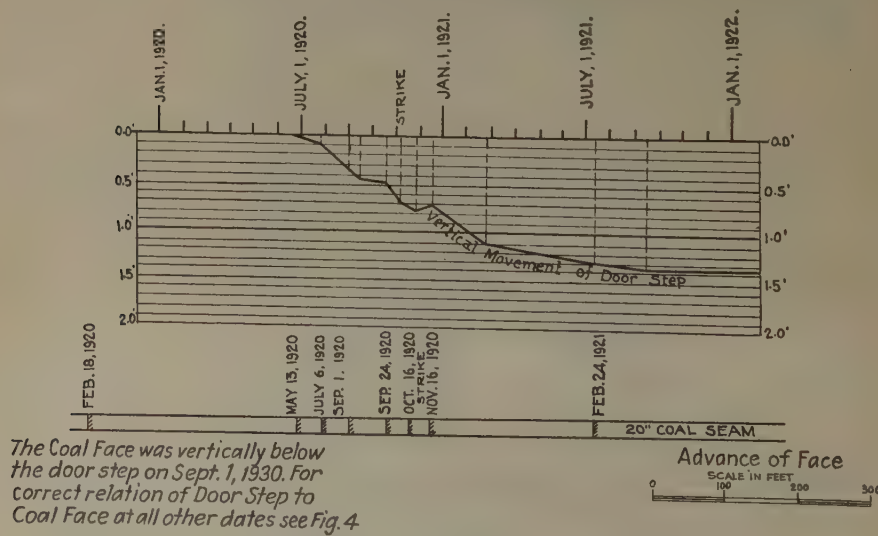


FIG. 13.—MOVEMENT OF POINT 27 IN RELATION TO TIME AND COAL FACE.

which prevented regular subsidence. For these reasons it is unlikely that on complete settlement of the surface the tower would have returned to original vertical position.

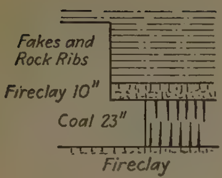


Fig. 14.

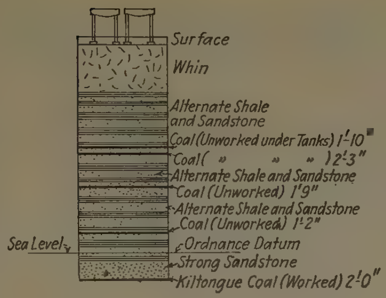


Fig. 15*a*.

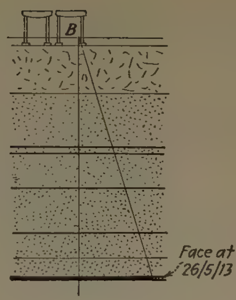


Fig. 15*b*.

Fig. 14.—Section of Kiltongue coal seam.

Fig. 15.—Kiltongue coal workings underlying water tanks.

a. Strata above seam. *b*. Section on line MN of Fig. 16. Angle of draw, 17° 40′.

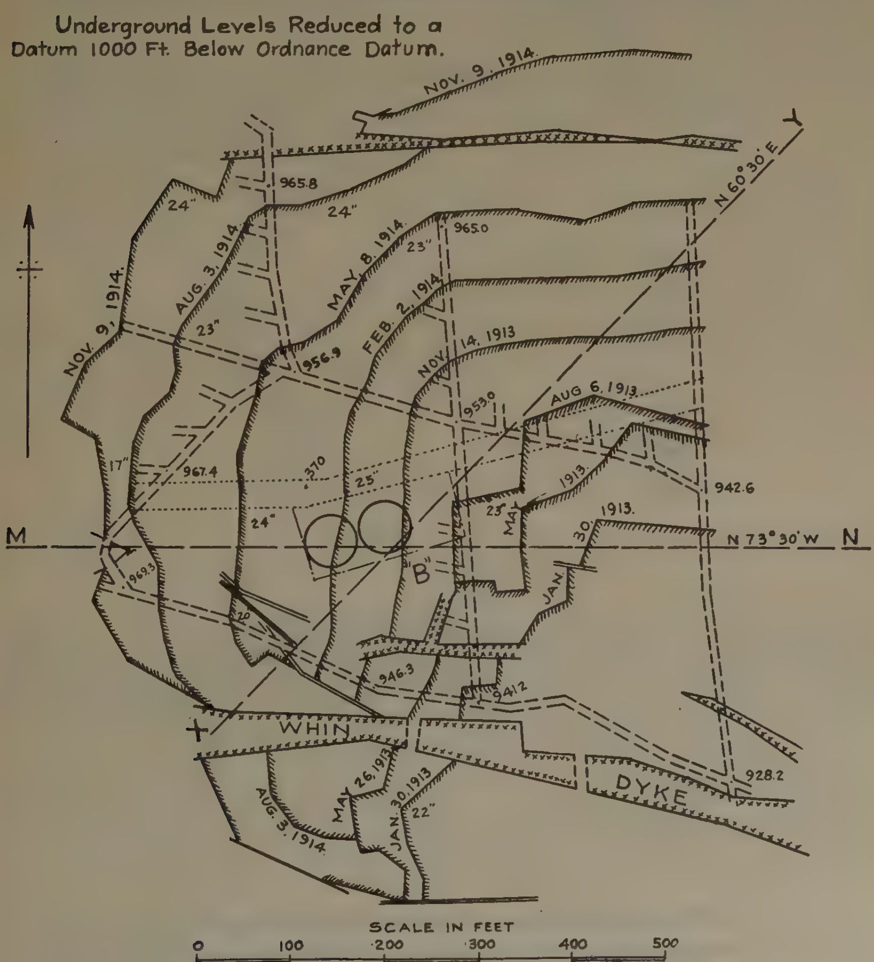


Fig. 16.—Plan of Kiltongue coal workings underlying water tanks.

The position of plumb lines A, B and C is given with movements seen in Fig. 19 and the measurements in Table 2. The elongation of line AB to

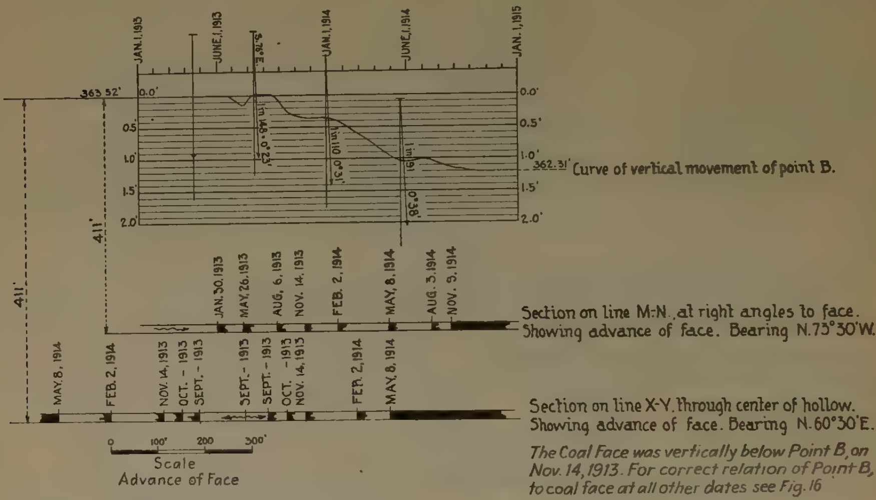


FIG. 17.—MOVEMENT OF POINT B OF FIG. 16 IN RELATION TO TIME AND FACE.

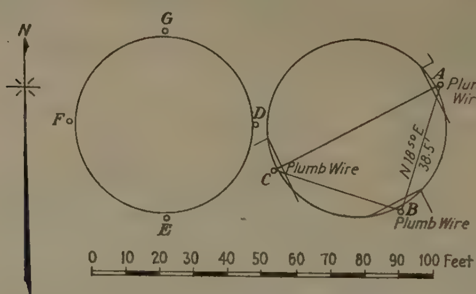


FIG. 18.—POSITION OF LEVEL POINTS AND PLUMB WIRES ON WATER TANK. LENGTH OF PLUMB WIRE, 40 FEET 2½ INCHES.

Variation in Level

	May 15 1913	June 16 1913	July 24 1913	Aug.12 1913	Sept.13 1913	Oct.16 1913	Nov.20 1913	Jan.6 1914	Mar.10 1914	May 28 1914	June 23 1914	July 1914	Sept.15 1914	Oct.27 1914	Total Fall May,1913 to Oct.1914
A	363.52	363.58	363.38	363.60	363.54		363.20	363.37	363.03	362.64	362.67	362.62	362.48	362.40	1.12
B	363.52	363.54	363.37	363.56	363.54	363.25	363.17	363.26	362.87	362.47	362.52	362.50	362.39	362.31	1.21
C	363.23	363.30	363.08	363.27	363.26		362.88	363.92	362.63	362.22	362.27	362.29	362.19	362.12	1.11
D	362.30														
E	362.23	362.27	362.11	362.27	362.27		361.99	362.12	361.83	361.45	361.47	361.47	361.34	361.27	0.96
F	362.56	363.62	362.41	362.66	361.59	362.35	362.30	362.49	362.25	361.88	361.88	361.81	361.65	361.55	1.01
G	364.18	364.24	364.01	364.25	364.19	363.93	363.88	364.07	363.76	363.37	363.37	363.30	363.14	363.05	1.13

A' B' on Fig. 19, etc., is a direct measurement of the distortion of the upper structure between the points A and B. Had this been rigid, cracks of an aggregate width equal to the difference would have been observed.

Wooden centers were put in and the upper structure bound with strong wire ropes drawn tight with screws. In fact, slight distortion of the circle took place without any damage being done to the structure.

These observations together with others not here referred to lead me to suggest that the nature of subsidence caused by the complete extraction of a seam of coal worked longwall and without interruption is a wave passing along the surface of the ground, the length of the wave being parallel to the coal face and the crest of the wave somewhat in front of the coal face; the slight rise that takes place before subsidence is of the order of 2 per cent. of the thickness of the seam and the total subsidence is of the order of 60 per cent. of the thickness of the seam.

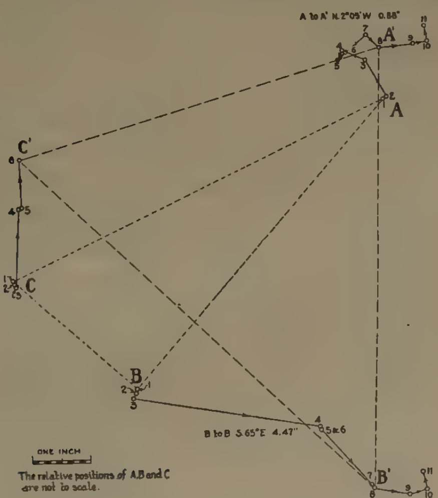
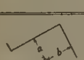


FIG. 19.—MOVEMENT OF PLUMB BOBS ON WATER TANK.

TABLE 2.—*Movement of Plumb Wires at Garthamlock (Kiltongue Workings)*

(Measurements in Inches)

			1	2	3	4	5	6	7	8	9	10	11
			April 15 1913	May 15 1913	June 16 1913	Aug. 12 1913	Sept. 15 1913	Oct. 16 1913	Dec. 29 1913	Jan. 6 1914	March 10 1914	May 19 1914	June 23 1914
	A	a	9.94	9.94	9.19	8.87	9.00	8.94	8.81	9.12	9.31	9.37	9.12
		b	4.62	4.69	4.62	4.34	4.19	4.37	4.81	4.94	5.50	5.75	5.81
	B	a	9.00	9.06	9.12	6.44	6.44	6.44	6.06	6.06	5.56	5.25	5.19
		b	4.62	4.69	4.75	7.62	7.69	7.69	7.94	8.00	8.37	8.44	8.12
	C	a	8.50	8.56	8.62	7.44	7.44			6.69			
		b	4.19	4.19	4.19	3.53	3.50			3.19			

The slope or angles made by the curve with a horizontal line vary with the thickness of the seam, the depth of the seam and the speed of advance of face. The angle in such a case is easily measured by hanging a plumb line.

The upper part of a wall built at right angles to the line of face is in tension and cracks will open while the curve is convex; then cracks will close or pavements on the surface will lift, or even the wall may bulge in the compression when the curve is concave.

Regard should be paid to these principles when laying out a coal face to pass below a heavy and valuable building.

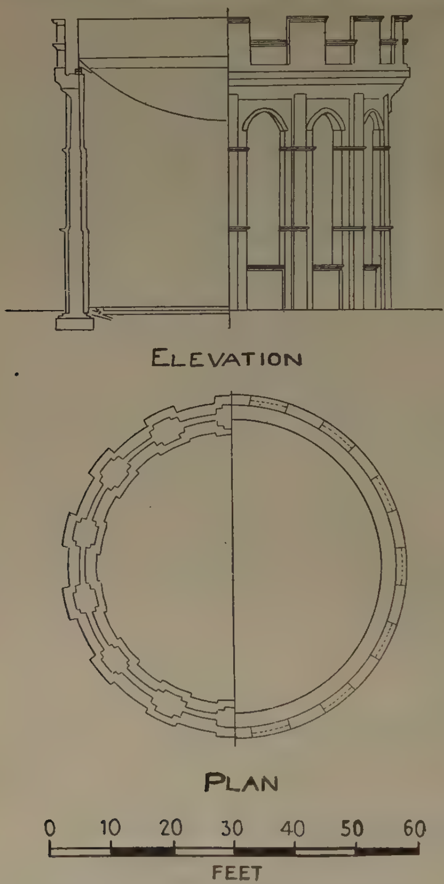


FIG. 20.—DETAIL OF WATER TOWER.

PRECAUTIONS AGAINST SUBSIDENCE

In the discussion of the Montz-Norris paper[3] reference was made to the effect of hydraulic stowing. We have not much experience of this in Scotland, as it is too costly. In this connection we have experience of water filling and when an old waste full of water is pumped out it is a common experience to find subsidence taking place on the surface as the water recedes.

I agree with the opinion expressed in the discussion of the paper by Montz and Norris that more damage is done to surface buildings by leaving small pillars below them than by complete extraction. To be safe, a common rule here is to leave a pillar extending a distance in every direction around a building equal to one-half the depth.

[3] H. W. Montz and R. V. Norris: Subsidence from Anthracite Mining. *Trans.* A. I. M. E., Coal Div. (1930) 122.

Subsidence in the Sewickley Bed of Bituminous Coal Caused by Removing the Pittsburgh Bed in Monongalia County, West Virginia

By S. D. Brady, Jr.,* Morgantown, W. Va.

(Fairmont Meeting, March, 1931)

In Monongalia County, West Virginia, the Pittsburgh and Sewickley beds lie west of the Monongahela River and underlie practically all the western end of the county. The average thickness of the Pittsburgh bed is 8 ft., where it is being mined at present, sometimes running up to 9 ft. The Sewickley bed is 4½ to 6 ft. thick and lies 90 to 100 ft. above the Pittsburgh bed.

As the Pittsburgh bed had a known value, a large part of it was bought up without consideration of the Sewickley bed. When the Monongahela Railway was built up the west side of the Monongahela River, other interests bought up the Sewickley bed, as it was found to be equal in quality to the Pittsburgh, with slight local exceptions. The two beds, therefore, were owned by different people, and naturally each owner opened his mine and ran it without regard to the working of the other bed. Consequently, considerable portions of the upper bed were seriously affected for economical mining, especially on Scotts Run.

In recent years there has been a tendency, influenced by the decision of the West Virginia Supreme Court, on the part of the operators in Monongalia County to try to coordinate their projections in the layout of their mines.

A typical geologic section, according to the report of the West Virginia State Geological Survey on Monongalia County, agrees with Fig. 1. It is apparent that the rock above the Sewickley bed and between the Sewickley and Pittsburgh beds is of a rather friable nature, therefore it is easily broken in pillaring. About the only difference between this section and the geologic section in the Lowsville district is that in the latter district the rock between the coals is a denser sandstone and contains little shale and limestone and no evidence of Redstone coal. It is a laminated sandstone, and although it makes a harder mass, it is easily broken.

While working in the Lowsville district, the author had an opportunity to examine a neighboring mine in the Sewickley bed, which was over a section being pillared in the Pittsburgh bed. The two mines belong to dif-

* Superintendent, Osage Coal Co.

ferent companies, and no effort was made to correlate their development. The rooms in the Sewickley bed were driven on 50-ft. centers and were 300 ft. long, and the rooms in the Pittsburgh bed were on 90-ft. centers, 300 ft. long. The operator in the Pittsburgh bed developed his section last, on the best approved mining practice of driving up the room and pillaring it back immediately. The operator in the Sewickley bed had

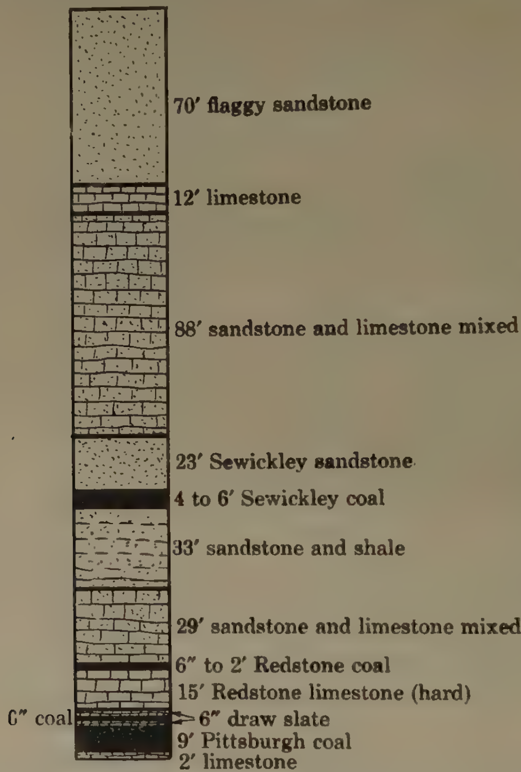


Fig. 1.—Typical geological section on Scotts Run, West Virginia.

the same idea, but after developing a number of rooms he decided to close the mine, and the operator in the Pittsburgh bed continued to operate. Later, when the operator of the Sewickley bed resumed operations he found it inadvisable to mine some of his coal, on account of the pillaring of the lower bed, so he proceeded to work in advance of the pillar line in the Pittsburgh bed, but as his mine was operated only intermittently the pillar line in the Pittsburgh bed caught up with his pillaring and passed it. The effect of the pillaring in the lower bed was indicated in the upper bed first by cracks in the pavement and a gradual subsidence of the pavement, which left the Sewickley coal hanging to the roof. This pillaring of the Pittsburgh bed was continued, and at times a space

of 18 in. or more was observed between the Sewickley coal and pavement. Some weeks later, the roof caved and ruined the section, so that further economical mining of the upper bed was out of the question. Sometimes caving occurred overnight and sometimes it took as long as two weeks. This mine has been practically abandoned, although mining was continued for almost a year before a squeeze across the main heads compelled its closing.

At a Sewickley mine, on Scotts Run, where the rooms were driven on 50-ft. centers, 300 ft. long, and where the rooms in the Pittsburgh bed were driven on the panel system with 40-ft. centers and 300 ft. long, one section in the Sewickley bed was partly developed when the robbing of the room pillars in the Pittsburgh bed developed a squeezed condition, causing a cessation of mining in the Pittsburgh bed. The effect of this squeeze was that overlying strata, including Sewickley coal, settled as a whole, dropping the Sewickley bed about 18 in. on the first day and a few more inches later, thereby forcing a cessation of mining in this section of the Sewickley bed, and no attempt was made to start up again for four or five years. Then, about 6 in. of roof slate was cleaned up from the roadway and mining was resumed. As the coal was not crushed, a recovery of 95 per cent. was reported. The only unusual difference in the mining was a greatly increased use of timber. The water disappeared through the cracks in the pavement. This unusual recovery was caused by the Pittsburgh bed being squeezed and the overlying starta settling as one mass.

At another plant in the Scotts Run field, the main headings in the Sewickley bed, consisting of two headings with 100-ft. barrier pillar on each side, were driven half on to bisect the property. Sections were fully developed on both sides with butt headings driven at an angle with the mains, and rooms were driven from them on 50-ft. centers and 300 ft. long. The sections on the right of the mains were pillared out, but on the left side were left standing. A rib line was started in the Pittsburgh bed underneath the pillared-out Sewickley section, which as it progressed crossed under the Sewickley main headings. The rooms in the Pittsburgh bed were driven on the panel system with 90-ft. centers and 300 ft. long. Here the officials in the upper coal first noticed cracks in the pavement and the disappearance of the mine water. Then the roof became bad and coal began to crush out of the ribs, and a squeezed condition developed in the Sewickley bed. Several weeks later, the roof caved one night, closing the headings. The section in which the rooms were developed on the left side of the mine, and not pillared, was left in good condition because it was situated over the Pittsburgh main heading and barriers. The new Sewickley mains were driven through these old rooms and have given no trouble. No plans were made originally to work these mines simultaneously because of different ownership.

At the Osage mine of the Osage Coal Co., situated on Scotts Run, the property map was studied with the idea of getting the largest recovery in both beds, and, in an economical manner, to work both beds. The Pittsburgh mine was laid out in conjunction with the Sewickley, so that pillaring could be started in the upper bed first. The Pittsburgh bed has an average thickness of 8½ ft. and the Sewickley an average of 4 ft. The cover over the latter varies from 0 to 150 ft., and the interval between the Pittsburgh and Sewickley beds is approximately 100 ft., as shown by levels.

Both mines were developed on the panel system with the rooms in the Pittsburgh bed on 90-ft. centers and 300 ft. long and the rooms in the Sewickley on 45-ft. centers and 300 ft. long, making all headings in the Sewickley overlie all headings in the Pittsburgh bed. Rooms in the Pittsburgh bed were fully developed before pillaring was begun. In the Sewickley bed, pillaring was begun in the southeast corner of the property in January, 1928, and after operation for three years, a pillar line 2200 ft. long is working and is recovering 98 per cent. of all possible coal. The pillaring of the Pittsburgh bed was started in the same corner of the property, about four months later, in April, 1928, and after three years, the Pittsburgh pillar line is, on an average, 100 ft. behind the Sewickley pillar line. No bad effect on either bed has been noticed during this simultaneous pillaring.

The writer has noticed large cracks from 6 to 12 in. wide on the surface overlying a pillared-out section after removal of all the coal in the Sewickley bed. These cracks appear at no definite interval but generally are parallel to the pillar line in the coal and apparently follow the pillar break at an offset from it of about 10 ft. for each 100 ft. of cover, thereby making the solid roof overhang the pillared-out section. For this reason, it has been possible to pillar the Pittsburgh coal at almost the same point as the Sewickley bed. After the Pittsburgh bed is pillared, the surface shows new cracks, and some of the earlier ones close up, and where there is only 100 ft. or less of cover over the Pittsburgh bed, the same conditions as to the angle of draw exist as in the Sewickley bed; that is, the cracks appear in the rear of the coal face.

The Pittsburgh pillar line is parallel to the Sewickley pillar line, and in the Pittsburgh bed there is no evidence of any additional weight, as shown by the absence of crushed coal or bad roof. In fact, we are getting about 10,500 net tons to the acre in the Pittsburgh bed, according to an accurate estimate by engineers, while in the Sewickley we are recovering about 7000 net tons to the acre, and there is also no evidence of squeezing in this bed.

A serious objection to this plan of mining is that in order to secure tonnage in the Pittsburgh bed it was necessary to drive all the rooms in the Pittsburgh bed and remove the tracks from them. Later it was

necessary to re-lay these tracks when pillaring was started. Some of this additional expense was offset by the fact that a number of the old rooms had fallen in, necessitating the driving of splits in the pillars where the falls were heavy. These splits have caused little trouble, owing to the original large room centers.

Conclusions

The conclusions to be drawn from this practical experience are that:

1. The Sewickley and Pittsburgh beds of coal can be mined simultaneously in an economical and successful manner, with average recovery.

2. The Sewickley bed can be removed before the Pittsburgh bed with no harm to the mining of the latter, if the Sewickley bed is completely and properly removed before the pillaring in the Pittsburgh bed is undertaken.

3. The Sewickley bed can be worked successfully, where the intervening strata is of a soft nature and the Pittsburgh bed has been completely removed, with the additional cost of extra timber.[2]

DISCUSSION

(*Thomas G. Fear presiding*)

T. G. Fear, Fairmont, W. Va.—In the Maryland field, where the Big Vein is almost completely mined out before the Tyson, about 90 ft. above, is worked at all, we are much interested in the subject of Mr. Brady's paper. The Big Vein there is even thicker than it is in West Virginia. It certainly is inconvenient to go to the trouble of setting up conveyors, advance about 50 ft. and find the bed entirely gone. With 8½ ft. of Big Vein below and only about 28 to 32 in. of Tyson, it does not take much subsidence to take the Tyson away entirely.

J. D. Sisler, Morgantown, W. Va.—I have come to the conclusion that coal beds can be mined at almost any interval. In one instance the Pittsburgh coal was mined out, about 1882, the coal being 8 ft. thick, and when it was mined the Redstone, which lies only 12 to 15 ft. above, was considered entirely lost. The Redstone coal was poor quality and was thought to be noncommercial. The stumps were never pulled in the Pittsburgh bed. About 15 years ago, when the main body of the Pittsburgh bed had been removed, operators had to seek elsewhere for a supply of coal and they turned to the Redstone. It has been mined successfully for the past 10 years with a recovery of 50 per cent.

Another example is the Punxsutawny region of Pennsylvania. The lower Freeport was mined out, the E bed was considered entirely lost and no provision was taken to protect the bed above it. The result was that the D coal was badly fractured and some of it was lost. By careful mining methods it was possible to recover 65 to 70 per cent. of the upper bed. The interval between these beds ranges from 45 to 60 feet.

I have come to the conclusion that it is possible to mine almost any bed, providing there is an interval of at least 10 or 12 ft. between the beds.

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[2] For corroboration, see H. N. Eavenson: Mining an Upper Bituminous Seam after a Lower Seam has been Extracted. *Trans.* A. I. M. E. (1923) **69**, 398.

H. N. EAVENSON, Pittsburgh, Pa.—Some years ago, while in the southern part of the state, I had occasion to look into this matter, and I think that the conclusions Mr. Brady has given are correct. At that time we examined a great many places in Pennsylvania, among others the one Mr. Sisler has mentioned. An interesting question arose about three years ago, concerning a limestone mine in Pennsylvania, which was being worked above a coal seam about 35 to 40 ft. below it. The coal was 40 in. thick and the limestone bed about 20 ft. The question of subsidence in a limestone bed is an entirely different problem from that in a coal bed, partly because of the great thickness of the bed and partly because it is almost impossible to tell how it will shoot when it has been cracked by subsidence.

In connection with that case, we visited the mine of the Marquette Cement Manufacturing Co. at La Salle, Ill. In 1915, that company was working a bed of limestone from 12 to 18 ft. thick, over a coal bed about 4 ft. thick about 400 ft. below it. The coal was being mined longwall, and at that time there was little knowledge of subsidence in that particular section. The coal company did not have proper mining rights, and, as a great deal of damage was done to the limestone, a suit was brought and the limestone company obtained an injunction against the coal company.[3] When we visited the mine, which was about seven or eight years after the mining of coal had ceased, the evidences of cracking and subsidence in the limestone mine were considerable in the older workings, but it was interesting to see that in some new sections opened in the limestone mine after the coal mining had stopped the cracks in the limestone 4 and 5 in. thick had filled up, and that the limestone was being mined satisfactorily. This was extremely interesting, in view of the different characteristics of the materials; and I believe it would be safe to say that almost any upper stratum can be mined after the lower one has been exhausted, and the better the mining has been done below, the easier will be the working in the top bed.

In regard to Mr. Sisler's statement about mining any bed provided there is 10 or 12 ft. between beds, I would say that this is true if such beds are not mined simultaneously. I think it would be almost impossible to mine the upper bed and try to maintain that mine while the lower bed was being taken out below it; but if the lower bed were taken out first and the strata given time to settle, there would be little trouble with the upper bed.

M. L. O'NEALE, Fairmont, W. Va.—Where the lower bed is gassy, will there be any extra hazard in working the upper bed, if the upper bed is considered a non-gaseous mine?

S. D. BRADY, JR.—When that occurs, West Virginia requires the upper bed to make the gas inspections. There has been one instance of gas in the upper bed coming from the lower bed, but in our mine there is no gas in either bed.

T. G. FEAR.—The most interesting method of mining that I have seen in this country is in Maryland in the Sonny mine. In the Big Vein, with 11 or 12 ft. of coal, originally between 5 and 5½ ft. of coal was mined out from the center of the seam. About two years ago, when people were mining that section, the roof had subsided and the seam could be seen, between 5½ and 6 ft. thick, with apparently a soft streak in the center composed largely of dirt and bug dust. Today it looks like a solid bed.

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[3] See page 27.

Introductory Notes on Origin of Instantaneous Outbursts of Gas in Certain Coal Mines of Europe and Western Canada*

By George S. Rice,† Washington, D. C.

(New York Meeting, February, 1931)

Instantaneous outbursts of gas in underground workings are similar in effect to great blasts of explosives, but without heat effects. Fortunately they occur only in a few localities in exceptional coal fields of the world and under exceptional geologic conditions. Such violent outbursts should not be compared with the ordinary outflows of gas into coal mines or with explosions of gas and dust. The phenomenon is essentially one of gas held in circumscribed areas under very high pressure and as a mining face or heading approaches such a place and the mining excavation sufficiently weakens the natural surrounding dam or shell, this is burst by the pressure just as by the discharge of a mass of explosives.

The outbursts, so far as known to the writer, have been confined to coal beds and have not occurred in other mineral deposits. Hence, as discussed later, he believes that it is the relation of the gases to the coal material that is a factor in conjunction with others in producing conditions which permit the phenomena to occur when mining is done.

The gases in most instantaneous outbursts have been hydrocarbon gases but in two coal fields they have been definitely found to be chiefly or wholly carbon dioxide. One of these localities in which CO_2 outbursts have been frequent and have caused loss of life is in the southern part of the Gard basin, near Alais, in southern France. In 1919, the writer visited the Fontaine mine in this field the day following the occurrence of an outburst by which two men were smothered by carbon dioxide and fine dust. After 24 hr. the CO_2 had not been entirely removed by the ventilating current. Occasional outbursts of CO_2 have also occurred in other small basins in France, causing accidents, as in Puy-de-Dome. However, mines in the Gard basin, north of those in which CO_2 outbursts have occurred, have experienced hydrocarbon gas outbursts.

In the Lower Silesian coal field, Germany, there is a group of mines which have been subject to many instantaneous CO_2 outbursts, some causing serious accidents, but the greatest disaster occurred recently

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* Published by permission of the Director, U. S. Bureau of Mines.

† Chief Mining Engineer, U. S. Bureau of Mines; Chairman, A. I. M. E. Ground Movement and Subsidence Committee.

and is described by P. A. C. Wilson, Dipl. Ing., in the following paper. At the time of a visit of the writer to the neighboring Neurode mine in Lower Silesia, in 1928, he discussed the outburst problems with Mr. Wilson and a member of the Lower Silesian Commission on Outbursts and studied the precautions used in these mines, which usually had been successful in preventing serious accidents from outbursts. In spite of the elaborate methods employed, there occurred a recent disaster at the Wenceslaus Colliery, in which 151 men were smothered. The Institute is fortunate in getting from Mr. Wilson, who took part in the rescue work, a general description of the conditions surrounding outbursts of carbon dioxide and a first-hand account of this disaster.

Instantaneous outbursts of hydrocarbon gases have been most numerous in certain deep mines of Belgium, with complex geological structure of the coal measures. The early outbursts in these mines were probably the first encountered in mining history. The very complete report of Stassart and Lemaire[1] enumerates 220 outbursts in the period between 1847 and 1891, of which 18 resulted in accidents, causing in all the death of 360 workers. This included the noted catastrophe at the Agrappe mine in 1879, in which 121 persons, or all those in the mine, were killed. Between 1892 and 1908 there were 137 instantaneous outbursts in Belgium, 33 of which caused accidents resulting in the death of 87 men.

Other mining districts in which hydrocarbon gases have caused great instantaneous outbursts with resulting loss of life have been in British Columbia in certain mines of the Crow's Nest Pass field, near Fernie, and in the Cassidy mine in the Vancouver Island field. Instantaneous hydrocarbon gas outbursts causing accidents have occurred sporadically in a few localities in South Wales, Scotland, England, and in a few other places in the world. Moreover, they have occurred in other parts of the two fields in which CO_2 outbursts occur—the Gard and Lower Silesia.

So far as is known to the writer, there have been no instantaneous outbursts in coal mines in the United States of sufficient violence to cause accidents to the personnel. However, as the conditions found in British Columbia may sometime be found in the mountain districts of the United States, it is well to be on our guard, although now the problem is of academic interest only for mine operators of the United States. Scientists, geologists and mining engineers do not limit their work to geographic boundaries so the origin of the gases, their migration through the ground and the way in which they are stored under high pressure is of interest to them.

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[1] Les Dégagements Instantanes de Grisou dans les Mines de Houille de Belgique (periode de 1892–1908) by Simon Stassart and Emmanuel Lemaire, Inspectors of the Belgian Mining Department, published at Brussels in 1910.

Instantaneous Outbursts of Gas Not Connected with Bumps

Instantaneous gas outbursts are frequently regarded as the same phenomenon as "bumps" and due primarily to high ground pressure. This is a theory still held by some who have not seen the different causes and effects. Bumps are like rock bursts in deep metal mines.

The writer, in 1917, was requested by the Minister of Mines of British Columbia to investigate bumps and outbursts in the Crow's Nest Pass coal field, following the occurrence of a disastrous one. In that investigation, the results of which were published in 1918,[2] he found that although the district was subject to both bumps and outbursts they had no relation to one another and did not occur at the same time or place or under the same conditions. In fact, there was surprisingly little gas given off after the greatest bump occurred, which threw down timbers, top coal and rash through an area of over 40 acres and caused an earthshake felt for miles away.

At the Cassidy mine, Vancouver Island, in the instantaneous outbursts of fire damp which the writer investigated in 1922, although large masses of coal were thrown out by the outburst gases there was no evidence of any rupture of the roof or severe shock. In other instances of instantaneous outbursts of gas in several countries, although the immediate roof may be soft and may be blown out or fall when the timbers are blown out, there have been no indications of bumps.

On the other hand, in the Springhill mine, Nova Scotia, where great bumps have occurred,[2a] no outbursts have accompanied the bumps, nor has the normal flow of fire damp or methane increased. In certain "open-light" mines in the mountains of eastern Kentucky, the outcrops of which are often above water level and no detectable amounts of gas are given off in normal operations, violent bumps have occurred, several of which were investigated by the writer, but in no case had gas been detected. In fact, open lights were used in recovery operations.

Other views than these appear to be held by certain mining men in Continental Europe, especially in France, who have considered that instantaneous outbursts are primarily due to high ground pressure locally causing the coal to burst and release occluded or absorbed gas. The writer cannot harmonize this view with the evidences in the separate phenomena but believes that the natural conditions in the one case preclude occurrence of the other.

Conjectures Regarding Origin of Hydrocarbon Gases in Outbursts

The writer, subsequent to making studies regarding outbursts which had occurred in British Columbia and southern France, reached the

[2] G. S. Rice: Bumps and Outbursts of Gas in the Mines of Crow's Nest Pass Coalfield. *Annual Report*, Minister of Mines of British Columbia (1917) 337–354.

[2a] G. S. Rice: Bumps at the Springhill Mine. *Annual Report*, Minister of Mines, Nova Scotia (1925).

conclusion which he expressed in a discussion that followed papers on instantaneous outbursts at the Ponthenry Colliery, South Wales, before the South Wales Institute,[3] that (1) the hydrocarbon gases of an outburst area were derived from the coal in that area in crushing by geologic thrusts that at the same time sealed up the outlets for escape of the gases; (2) that the pulverized coal dust has highly absorbtive properties, like charcoal although not to the same degree, and under pressure below 10 atm. absorbs volumes, almost in proportion to the pressure.

It has been observed and universally remarked by those who had studied outbursts that there was an enormous amount of impalpable coal dust which was blown out, together with broken coal. In some instances the amount of broken coal and dust is only a few tons, in other instances it amounts to several thousand tons.

When lumps of coal are crushed in a laboratory at ordinary room temperature they give off large volumes of occluded or adsorbed gas, chiefly hydrocarbon gas. This does not include gas which may be contained in adjacent joints, faces, butts, partings or slips. Some measure of the amount of gas released by crushing was obtained in a study made in the Crow's Nest Pass field in 1917, by the writer, which was given in the report to the Minister of Mines of British Columbia.[4]

Samples were gathered in certain mines of this field, sealed in containers while in the mines and sent to the Pittsburgh laboratory of the Bureau of Mines. They were crushed *in vacuo* at room temperature and the gases analyzed and measured by A. C. Fieldner, then chief chemist of the Bureau of Mines. Similar samples were gathered in certain mines of Vancouver Island and also one for comparison from the Pittsburgh bed in the Experimental Mine which has a shallow cover.

The volumes of gas varied widely in the samples from the different mines from 49 c.c. per 100 g. of coal from the Experimental Mine to 200 c.c. from the coal from the Coal Creek mine No. 3, Crow's Nest Pass field. The composition also varied widely. The ethane exceeded the amount of methane in the Coal Creek sample and formed large proportions of the gases of the other samples. The amount of carbon dioxide in the Coal Creek samples was 6 per cent. of the entire gas.

Graham's Experiments on Gas Solubility in Coal

J. Ivon Graham, chemist of the Doncaster Coal Owners laboratory, England, carried on tests on the solubility of the different gases in pulverized (200-mesh) coals.[5] He found a wide variation between different

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[3] G. Roblings: Outbursts of Gas and Methods of Working Seams of Coal Liable to Them. *Proc.* So. Wales Inst. of Min. Engrs. (Nov. 9, 1926). Discussed by George S. Rice and others, *Ibid.* (1929) **43**, 32–47.

[4] G. S. Rice: *Op. cit.*

[5] J. I. Graham: Permeability of Coal to Air or Gas. *Trans.* Inst. Min. Engrs. (1916–17) **52**, 338.

coals in the absorption of different gases, varying in the case of methane from 134 c.c., at atmospheric pressure, per 100 g. of cannel coal to 252 c.c. for Welsh steam coal. He also found that the relative solubility of nitrogen and carbon dioxide in one of the same coal-dust samples of 100 g., measured in cubic centimeters absorbed was nitrogen 58 c.c. and carbon dioxide 800 c.c. at atmospheric pressure.

Briggs' Tests of Gas Solubility of Ponthenry Outburst Coal

Prof. Henry Briggs[6] made tests on the absorption of methane, or fire damp, and carbon dioxide by dried "outburst" coal (size not stated but evidently dust) from the Ponthenry colliery, South Wales, under different pressures, also the absorption of methane by normal pulverized anthracite, which was a little less absorptive of methane than the outburst coal. He gives his results in a graph as calculated to cubic feet of gas "discharged" per cubic foot of solid coal. From the graph are derived these approximate figures.

Ponthenry Outburst Coal, Discharge of Gas per Cubic Foot of Solid Coal

Atmospheres (gage press)	2	4	6	8	10
Fire damp (98 per cent. methane), cu. ft.	3.5	5.5	7	7.5	
Carbon dioxide, cu. ft.	7	11	13	14.7	16.5

From this, under the conditions of Professor Briggs' tests, the absorptive effect for CO_2 is about double that of methane for the same pressure. It seems probable that if the pressures had been carried up to the critical liquefying pressure of carbon dioxide, at the temperature of the test, the curve of quantity rate of carbon dioxide absorbed by the powdered coal would have gone up still more rapidly than the quantity rate of methane until the saturation point was reached.

One of the most suggestive comments made by Professor Briggs in his paper is that "the coal ejected by these outbursts from the solid is dry," which leads him to suggest forcing water into the coal face to displace the gas.

The writer of this paper agrees, from his investigation of outburst places soon after outbursts have occurred, that the dust and broken coal are very dry, but he does not think that the coal from which the burst occurs is solid. As previously mentioned, he believes that part of the coal in these outburst places has been previously more or less crushed by geologic movement, to an impalpable powder, which fills spaces between partly crushed and broken coal, as indicated by the large volume of impal-

[6] H. Briggs: Characteristics of Outbursts of Gas in Mines. *Trans.* Inst. Min. Engrs. (1921) **61,** 119.

pable powder ejected. It is further conjectured that the dust and broken coal thus made locally in the coal bed in past ages have doubtless been more or less recompressed but not recemented together by the general overlying weight of the strata.

It has been a matter of general comment by investigators of outbursts that they originate in "soft" coal, quite generally surrounded by hard densely compacted coal without cleavage or normal joint planes. In fact, it seems probable that the fine dust acts as a sealing agent in the adjacent surrounding coal walls just as mud will seal the suction strainer of a pump. This sealing may also be a factor in the most puzzling feature of the phenomena that boreholes, drilled into the coal face, rarely seem to be effective in draining off high-pressure gas, which fact has made most of the practical mining men question or denounce the merits of exploratory boreholes required by most mining laws as a means of detection.

Tests by Lower Silesian Outburst Commission

In 1927, the Report of the Commission on Instantaneous Outbursts of Carbon Dioxide in the Lower Silesian Coal Field,[7] was published by the Prussian Mining Department. It is the most complete report on CO_2 outbursts that has been made covering geologic conditions, details of Lower Silesian outbursts up to 1926 and experimental work. Tests were made of the absorption and adsorption of CO_2 for 11 samples of coal and one of activated charcoal. The results of these tests (made by Prof. Dr. Ing. Otto Ruff of the Institute of Technology, Breslau) are summarized in Table 1.

The following general conclusions (freely rendered) were reached regarding A, B and C of the table:

A. (1) Lower Silesian coal absorbs no more carbon dioxide gas than other coals, for example, Upper Silesian coals; (2) the capability of absorbing CO_2 does not differ much (*a*) for hard or soft structure coal of the same bed, or (*b*) for powdered coal more than for lumps, or (*c*) for fine coal dust characteristic of outburst coal.

B. The amount of carbon dioxide absorbed into the pore spaces of the coal is secondary to the adsorption or solution phenomenon.

C. The amount of carbon dioxide released from coal at 2 atm. in the laboratory tests was sufficiently great for the explanation of volumes of gas released during an outburst.

The present writer suggests that the data given by the tests do not seem to support the view of A(2*b*) or A(2*c*); namely, that fine coal dust does not absorb more CO_2 than lump from the same coal bed. In one case of the data given, lump coal did not absorb more than one-half as much CO_2 as the dusts from the same coal.

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[7] Untersuchungen über die Entstelung und Bekämpfung der Kohlens äureausbruche im niederschlesischen Steinkohlenbezirk. Berlin, 1927.

TABLE 1.—*Absorption of CO_2 for Various Gage Pressures in Cubic Meters* (=35 Cu. Ft.) Calculated at Atmospheric Pressure per Ton (2204 Lb.) of Coal at 20° Centigrade

Sample No.[a]	Porosity Volume	1 Atmosphere[b]			2 Atmospheres			3 Atmospheres		
		A	B	C	A	B	C	A	B	C
1	0.102	2.4	0.1	2.5	4.	0.2	4.2	5.8	0.3	6.1
4	0.056	2.6	0.1	2.7	4.5	0.2	4.7	7.1	0.3	7.4
6	0.050	3.0	0.1	3.1	5 2	0.2	5.4	8.6	0.3	8.9
12				20.0			35.0			55.0

		6 Atmospheres			8 Atmospheres			10 Atmospheres		
		A	B	C	A	B	C	A	B	C
1	0.102	7.0	0.6	7.6	8.1	0.9	9.0	9.3	1.0	10.3
4	0.056	9.3	0.4	9.7	11.0	0.4	11.4	12.1	0.5	12.6
6	0.050	10.9	0.3	11.2	12.4	0.6	13.0	13.7	0.6	14.3
12				70.0			83.0			94.0

[a] 1 = borehole cuttings from the Roschen coal bed in the 4th level, where coal gives off much CO_2 but as yet no outbursts.

4 = coal sample from the Wenceslaus colliery from the Wilhelm coal bed giving off much CO_2 and where conditions threaten the occurrence of outbursts.

6 = coal sample from the Wenceslaus colliery, Wenceslaus coal bed, giving off little CO_2.

12 = activated charcoal.

[b] Atmospheres are indicated by A, chemically absorbed (Chemisch gebunden); B, held in pores; C, total.

Apparently the investigators did not consider that much higher pressures may be attained under mining conditions than 10 atm. or about 150 lb. per square inch. Structural folding may produce enormous local pressures and the direct pressure of only 1000 ft. thickness of superposed strata would be sufficient to liquefy CO_2 if held tightly, and liquefaction would greatly increase the amount of CO_2 that might be held in a unit volume of coal.

H. P. Greenwald's Diffusion Tests through Coal Dust

At the request of the writer of this paper, H. P. Greenwald, physicist of the U. S. Bureau of Mines, began a preliminary study, in 1927, of the diffusion of gases through highly compressed coal dust. This investigation had to be suspended because of pressing official work. Some of Mr. Greenwald's notes are as follow:

The dust used was from a low-volatile coal from central Pennsylvania, contained 19 per cent. volatile matter on a moisture and ash-free basis,

and 97 per cent. thereof passed a 200-mesh sieve. It was pressed into a steel tube of 1.5 in. internal diameter at a pressure of 20,000 lb. per square inch. The tube contained 14.1 oz. of dust, the compressed length of which was 11.92 in.; this gave a specific gravity of 1.26, close to that of coal in place. After some preliminary tests the passage of a mixed gas through this dust was observed over a period of 19 hr. The gas was confined in a large-sized oxygen cylinder at a pressure of 45 atm. and passed through a small copper pipe to the tube in which the dust was

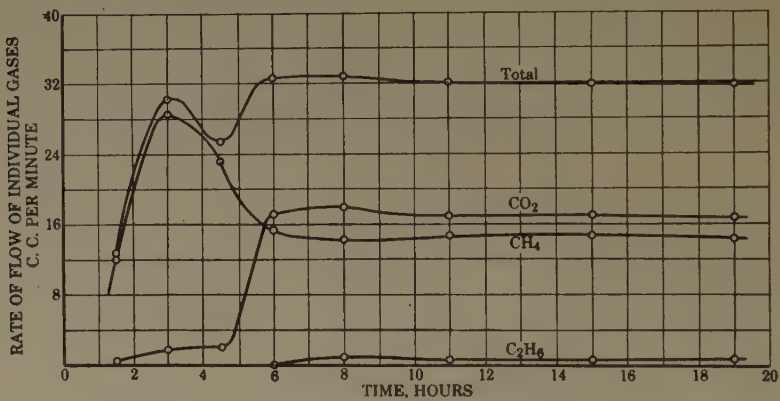

FIG. 1.—RATE OF FLOW OF PRINCIPAL COMPONENTS OF MIXED GAS THROUGH COMPRESSED COAL DUST.

pressed. The original composition of the gas and that effluent from the compressed dust are given in Table 2. Sample 0 is the original gas and the numbers of the other samples designate the time in hours after the start of the experiment at which they were collected.

TABLE 2.—*Analyses of Gases by Laboratory of U. S. Bureau of Mines*
Samples Numbered by Hours Elapsed

Sample No.	0	1.5	3	4.5	6	8	11	15	19
Constituents									
CO_2	51.3	2.9	5.2	7.7	51.3	53.1	51.8	51.5	51.8
CH_4	41.9	61.9	92.2	89.7	46.0	42.2	45.4	45.0	44.8
C_2H_6	5.0	0.0	0.0	0.0	0.3	2.9	1.3	1.6	1.5
N_2	0.2	25.3	2.4	2.4	2.2	1.8	1.4	1.6	1.8
O_2	1.6	9.9	0.2	0.2	0.2	0.2	0.1	0.3	0.1

The rate of flow of total effluent gas was measured continuously during the experiment and the flows of the three principal components were computed from this and the analyses of Table 2 were plotted in Fig. 1. There is considerable excess nitrogen and oxygen in sample 1.5, which

was being driven out of the coal by the compressed gases. When these are removed the sample contains 93 per cent. methane.

The behavior of the gases passing through the compressed coal dust was doubtless complicated, but after a steady state was reached, that is, after 10 hr., was probably mainly diffusion, the laws of which have been extensively investigated. If only diffusion takes place the compressed dust may be likened to a series of porous membranes and the rate of passage of a gas through such a membrane depends on the difference of pressure of that gas on the two sides therefore, and varies inversely as the square root of the density of the gas. Mixed gases behave independently of each other, each gas acts as though the others were not present and diffuses at a rate determined by its own partial pressure and density. One can then state that the relative rates of diffusion of the CH_4, CO_2, and C_2H_6 through the coal dust should be as 100:74:9 provided the composition of the gas entering the coal remained constant and none of it was absorbed by the coal.

Discussion of Diffusion Tests

The compositions of the original and effluent gas mixture at 15 hr. in the Greenwald tests are as follows:

Constituent	Original Gas, Per Cent.		Effluent Gas, Per Cent.	
CO_2	51.3		51.5	
CH_4	41.9	46.9	45.0	46.6
C_2H_6*	5.0		1.6	
N_2	1.6		1.6	
O_2	0.2		0.3	
	100.0		100.0	

* Includes small amounts of higher paraffin hydrocarbon series.

The CO_2, N_2 and O_2 are practically the same in the original and effluent gas; C_2H_6 is much lower and CH_4 higher in the effluent gas and their sum is practically the same in the original and effluent gas.

This last result is significant, although it requires caution to accept unqualifiedly without further tests. It would appear that a certain amount of ethane has replaced and released an equivalent amount of methane from the coal. It has long been a matter of wonder why it is that, although a large proportion of the hydrocarbon gases, obtained from coals crushed *in vacuo*, is made up of the ethane and higher paraffin hydrocarbons, it is extremely rare when traces of the ethane series are found in the mine air of coal mines; the above conjectured interchange of gases in the coal may account for this.

The foregoing tests of the diffusion through compressed coal dust of the gases of the synthetic mixture in the order of density of the individual gases (except ethane, which lagged but which was in too small a proportion for final acceptance) appear to confirm the writer's earlier hypothesis that the "core" of individual outburst areas is perhaps filled with the denser gases; *viz.*, carbon dioxide and the ethane-propane series. It is suggested that these gases under high pressure from geologic compression are in a liquid phase, into which enters also the effect of the phenomenon of surface adsorption. The conjectured pulverization of a part of the coal in a future outburst area provides an enormous amount of surface for such adsorption effects.

It is further thought that in many cases the methane and nitrogen of the normal occluded or contained gases of coal have more or less diffused and escaped from the area in the course of geologic ages, leaving the carbon dioxide and heavier hydrocarbon gases behind.

If this is so, the more easily liquefied gas, carbon dioxide, would in many cases be the principal outburst agent when the approaching mine excavation has sufficiently weakened the walls.

Liquefied CO_2 suddenly released from a steel container is being used as a blasting agent in some gassy coal mines of the United States and England as a substitute for the usual explosives.

It is not thought that all methane has disappeared from such outburst areas. None of it may have escaped, but the foregoing test data suggest that the gas which bursts out may be a mixture of residual gases in various proportions, varying according to the degree of escape of lighter gases through the enclosing coal walls.

The carbon dioxide as found in the Gard and Lower Silesia mines undoubtedly originated in the intrusive igneous rocks below or near by and has reached the coal beds through faults or extensive joint planes of structural folds. It is suggested that part of the gas has "entered" in local areas where the coal has been more or less crushed to dust and has been held, with more or less hydrocarbon gases from the pulverized coal, as by previous hypothesis, possibly in a liquefied state.

If these several hypotheses are accepted it would account for the varying composition of the gases adjacent to outburst areas.

The carbon dioxide outburst commission of Lower Silesia proposed, as a means of giving warning of an impending outburst, the boring of holes to a depth of about 2 meters (6½ ft.) in the face of the working place, which might afterwards be employed for blasting but first used for obtaining samples for analysis of the gases coming off, the percentage of CO_2 present to be the index of the hazard. From experimental tests in outburst areas, it was concluded that when the percentage of carbon dioxide rose to 50 per cent. there was danger and when the percentage

rose to 75 per cent. the danger was so imminent that men should be withdrawn from that district.

In 1928, the writer was informed by one of the local mining engineers that the method of obtaining a warning had not been found reliable. It is possible that this is because of the varying composition of gases held in the outburst "area" involving a question of prior diffusion and, on the other hand, the borehole sample may contain carbon dioxide from some fault or joint plane which was not under pressure, because of free drainage elsewhere into the mine.

Summary of Writer's Views on Instantaneous Outbursts[8]

1. They occur only in coal-mining operations.

2. They occur only when there has been profound geologic movement after formation of the structure of the coal (which has reached the bituminous stage or higher—the Ponthenry, South Wales, outbursts occurred in semi-anthracite[9]), the movement having crushed the coal locally.

3. That the crushing of the coal liberates 2 to 5 volumes (at atmospheric pressure) of mixed gases, methane, ethane, etc., nitrogen and carbon dioxide per volume of coal.

4. At the time the geologic thrusts took place, the coal beds in all the present outburst localities were under deep cover, so the gases in these localities were locally retained under pressure. During subsequent ages the cover was extensively removed by erosion and in some of these localities the least dense gases, methane and nitrogen, diffused, but left in other instances, concentrations of carbon dioxide and the denser hydrocarbon gases.

5. That in Gard (France), Lower Silesian fields and a few other places CO_2 was derived from igneous intrusions below, which escaped into the coal beds through fault or joint planes but were similarly held in place.

6. That pulverized coal is akin to charcoal in its absorptive power for gases, particularly carbon dioxide.

7. That carbon dioxide in greater or less proportion, in outburst gases through the fact of its low liquefaction pressure (about 1000 lb. per square inch at ordinary mine temperature), is perhaps the important factor of outbursts in the Gard and Lower Silesia, and may also be an important factor in outbursts of fire damp. The latter also occur in both the districts named as well as in Belgium, British Columbia, and occasionally in Great Britain and a few other countries.

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[8] Originality of these views is not claimed except as to paragraphs 3, 4 and 7; and as concerns the latter it is not intended to suggest that very high gas pressures always exist.

[9] American classification based on volatile-combustible ratio.

Remedies for Outbursts

Apparently there is no absolute way of preventing dangerous instantaneous outbursts by adoption of certain mining methods. Theoretically, advancing longwall in a continuous face would be most likely to drain the gases slowly, but practically it is impossible to employ advancing longwall in a continuous face, because in all mines where outbursts occur the coal measures are folded and faulted and either dipping or heavily "rolling," so that the best that can be done is to block out by narrow work or use, as is done in Belgium, an irregular stepped longwall. Most outbursts occur in driving narrow work, but they have also occurred in mines of Belgium and Lower Silesia in stepped or block longwall.

The general precautionary method required by the Government mine inspection departments in each of the countries affected is to bore exploratory holes about 6 or 7 ft. deep in the face. Mine operators in some coal fields, in mines where no blasting is permitted, as in British Columbia, think exploratory holes are a needless expense and do not discover outbursts. However, how many outbursts may be prevented by slow bleeding of the gas through boreholes is a question.

The French and Upper Silesian authorities, and recently the British in the Ponthenry colliery, permit "blasting off-the-solid" by distant electric firing when all men are out of the mine (France) or from places of safety (Lower Silesia and South Wales) and do not permit undercutting or picking into the coal face. The British Columbia and the Belgian inspection departments do not permit blasting in coal in mines subject to outbursts on account of the possible hazard of igniting gas or coal dust.[10]

Conclusions of the Writer as Regards Precautions

The writer's tentative conclusions as regards precautions are:

1. There should be preliminary blocking out of the ground by levels and raises, which appears to be the most practicable method under the usual natural conditions in outburst mines.

2. Exploratory holes should be drilled in the working place in the middle part of the bed, at least two in each heading, respectively, close to and parallel with each rib, the holes to be not less than 20 ft. long. The drilling should be done by use of strongly built machines wedged in place and so arranged that men would never be working directly in line with the hole when drilling.

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[10] Professor Denoël, of the University of Liége, writes that shock blasting is allowed by special permission in Belgium, under the same precautions as in France.

3. If a hole strikes a strong flow of gas, drilling and other operations in the heading and neighboring places should cease until the pressure is relieved.

4. If the holes strike no gas under pressure in mines subject to hydrocarbon outbursts, after thoroughly rock-dusting the heading, if other conditions warrant, the holes may be heavily charged with a permissible detonating type of explosive, stemmed with rock dust and fired electrically from outside the panel guarded with safety doors or from the outside of the mine when all the men have left the mine, as is done in the Gard field of France and in all coal mines in the State of Utah.

If the blasting of these large charges does not induce an outburst, it seems reasonably safe that the work could proceed by the usual coal-mining methods and exploratory holes started or continued in line with the previous holes.

5. As an additional precaution, the division of the mine into districts by safety doors, such as are employed in the Ruben mine, Lower Silesia, is an excellent plan.

6. If each district of a mine can be completely isolated by safety doors when a threatening condition occurs, as soon as the men in that panel are checked out by the district foremen, this measure, in addition to that of long exploratory holes in the headings which block out the ground, followed by heavy blasts fired from a place of safety or from the surface, would seem to give as much precaution as can be taken under the difficult conditions.

7. It is possible that the geophone might be useful in giving warning, especially if one might be devised to give warning automatically and safely and to be selective as to ground waves of a frequency such as those produced by gas moving through the coal under high pressure.

Although the use of exploratory holes has been practiced in the various outburst localities, the holes have not generally been long enough, and as yet, so far as is known, the proposed method of using very heavy localized blasts in these long holes to crack the ground as in blasting oil-well holes, has not been tried. The holes, as mentioned, would be bored in the middle or lower part of the coal to avoid as far as possible injury to the roof in blasting.

[For discussion of this paper, see page 102.]

Instantaneous Outbursts of Carbon Dioxide in Coal Mines in Lower Silesia, Germany

By P. A. C. Wilson,* Neurode, Germany

(New York Meeting, February, 1931)

Instantaneous outbursts of carbon dioxide in coal mines have occurred in Germany only in one part of the Waldenburg-Neurode mining district.[1] This mining region comprises the northeastern fold of the "Innersudetian" coal basin. The basin (Fig. 1) is in the form of a geologic trough approximately 60 km. (37 miles) long and 30 to 35 km. (19 to 22 miles) broad and is enclosed on three sides by ancient truncated mountain ranges—the Riesengebirge, the Bober-Katzbachgebirge and the Eulengebirge. Only the southeastern part of the trough is not enclosed by mountains of older rock formation.

The floor of the basin before the building up of the coal measures was formed on the western, northwestern and northeastern borders by the material from Archean rocks (gneiss and mica schists); in the northern part over this was deposited the Silurian formation. The carboniferous period is represented both by the Lower ("kulm") and Upper Carboniferous, with an unconformity between the stages; that is, the kulm was tilted up and partly eroded before the Upper Carboniferous was deposited. The kulm is a salt-water formation (with marine fauna) but Upper Carboniferous rocks are nonsaline water deposits, with fresh-water flora only. The Lower Silesian coal measures were thus formed in a lime basin fed by the running water coming down from the mountains.

As already indicated, the bottom series of the Upper Carboniferous formation, termed "Waldenburg strata" were laid down unconformably on the Lower Carboniferous. Over the Waldenburg strata there are the "Weisstein strata," an intermediate barren stage without coal beds, 400 to 500 m. (1300 to 1600 ft.) thick. Another unconformity followed, which is overlain by the "Saarbrucken" series, which contain the upper coal measures, and the "Ottweil" strata. In the last named, it is only on the southwestern fold that beds of coal are developed.

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* Mining Inspector of the Gewerkschaft Neuroder Kohlen- und Thonwerke.

[1] The standard publication on geology of Lower Silesia, the origin, holding and outbursts of carbon dioxide, mining and precautionary measures, tests in Lower Silesia and scientific opinions may be found in Untersuchungen über die Entstehung und Bekümpfung der Kohlensäureausbrüche im niederschlesischen Steinkohlenbezirk (1927) which contains articles by von Bubnoff, Kindermann, Prausnitz, Rademacher, Ruff and Werne. An article by von Bubnoff on the Neurode coal-mining district was published in *Fortschritte der Geologie und Palæontologie* (1930) **29.**

The number and quality of the coal beds vary greatly. Many beds of coal are less than 1 m. thick (40 in.); in thicker veins regular partings occur. In the lower coal measures there are 4 to 20 beds, in the upper coal measures there are 2 to 22 workable seams. The maximum total thickness of minable coal in workable veins, found at one place in the lower coal measures, is about 15 m. (50 ft.); in the upper coal measures it is about 35 m. (115 ft.). Beds of bituminous coal and of splint coal are alternately deposited. The rocks between coal beds are mainly conglomerate and sandstones, and to a lesser extent are shale. The geologically younger overlying strata are made up of lower new red sandstone (Permian), new red sandstone, chalk and recent formations. (Fig. 2.)

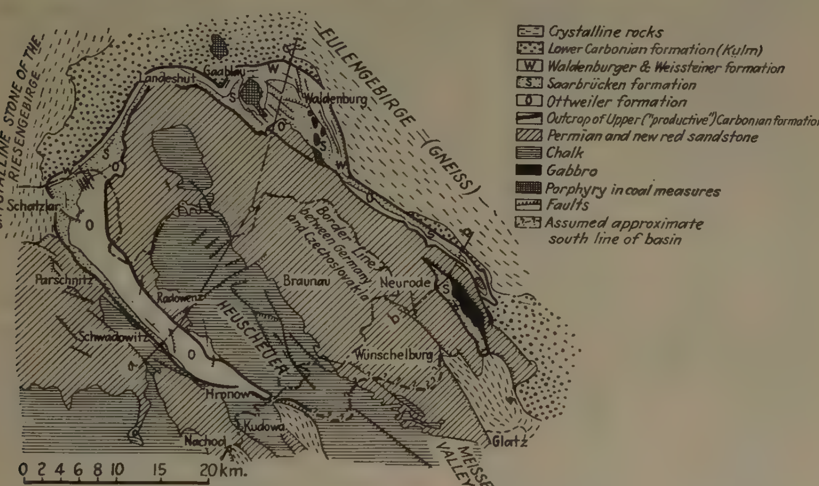

FIG. 1.—GENERAL MAP OF LOWER SILESIAN COAL BASIN—A BOG TYPE OF BASIN.

The Lower Silesian coal-mining district is distinguished by the widespread igneous rocks. The Neurode district is marked principally by a ridge of Devonian rocks which form a mountain that is about 8 km. (5 miles) long and 1.5 km. (0.9 mile) broad. It is composed of black (olivine) gabbro, green gabbro, serpentine and diabase. From these rocks have been derived by weathering a feldspathic material which is most valuable economically, a high-grade fireclay, which is found in beds 0.20 to 6 m. (8 in. to 20 ft.) thick. These beds underlie the lower coal measures. In them are sometimes found, in small amounts, spar iron ore, nickel pyrites, iron pyrites, lime spar, pharmacolite and pholerite and a certain amount of bituminous matter.

In the Ruben mine there are four workable fireclay beds and in the Johann Baptista mine two beds. None of the other mines in Germany is known to have workable fireclay.

In passing it may be remarked that these two mines, in addition to their coal production, produce about 120,000 tons annually of this very

high-grade fireclay, which is sold all over the world and provides 66 per cent. of the needs of all Germany.

The Waldenburg district is distinguished by the post-Carboniferous porphyry probably of Permian age. On both sides of the extension of the northeastern fold are found younger igneous rocks—basalts of the younger Tertiary epoch, and also diluvium of glacial origin. One of these younger volcanic centers is in the northwest, in the Bober-Katzbachgebirge, 23 to 35 km. (15 to 21 miles) from the Waldenburg district. The other center—probably of diluvian glacial age—is to be found near Landeck, 35 km. (21 miles) from Neurode. Both these volcanic districts are in the extension of a great disturbance, on which springs of hot water with carbon dioxide of Reichenau, Salzbrunn, Altwasser and Charlottenbrunn are to be found. In the Lower Silesian coal basin also there are hot springs containing carbon dioxide at Altheide, Reinerz, Kudowa and so on. So it is most probable that the carbon dioxide is of geologically recent origin. It is understood to be the secondary effect of the volcanic action, which took place in a recent geologic age, and it is probable that this throwing off of the gas is still going on.

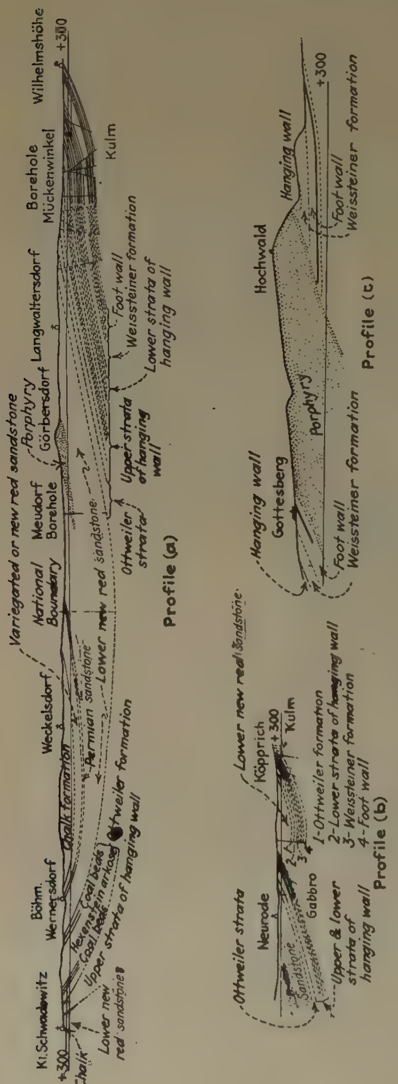

Fig. 2.—Profiles of Lower Silesian coal field.

All the district is affected by many geologic disturbances, which were caused principally by the profound movements in mountain building and took place in the various geological epochs in Lower Silesia that resulted from volcanic action. The faults and other effects of disturbances are to be regarded as the channels and spreaders of carbon dioxide coming up from depth.

The carbon dioxide tends to follow the faults and is not confined to any rock stratum. It may, in this way, cross a stratum in which the rock substance is nearly impervious. Therefore the existence of many

FIG. 3.—WORKINGS IN JOSEPH BED, RUBEN COLLIERY, SHOWING PLACES AT WHICH THERE HAVE BEEN INSTANTANEOUS OUTBURSTS OF CO_2.
Second stage level is 200 m. underground; third stage level, 300 m. and fourth stage level, 400 m.

faults crossing the rocks has permitted the carbon dioxide to spread over a wide area.

In other mining districts subject to instantaneous outbursts of gases, which generally are hydrocarbon gases, the idea has been put forward that

the origin of the gases may be due to the effect of sudden pressure. But this supposition does not appear to be correct, at least for the origin of the CO_2 in the Neurode district, as here the connection of igneous flows and intrusions of recent geologic age with the presence of carbon dioxide is clearly proved.

In this district the amount of CO_2 given off as the coal is mined varies not only as between one mine and another, but also considerably in the

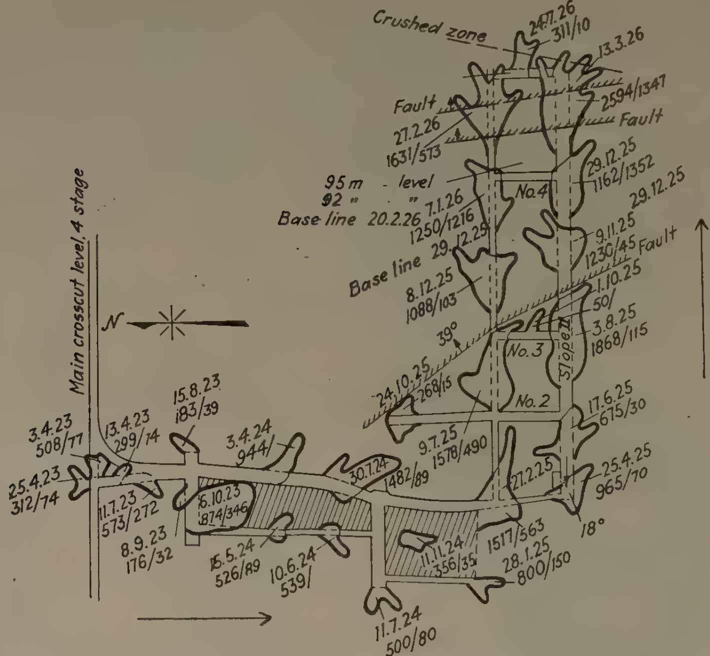

FIG. 4.—SERIES OF OUTBURSTS IN THE ANTON SEAM, RUBEN COLLIERY, FOURTH LEVEL. IN OPENING HEADINGS; DRIFTS INCLINED FROM APRIL 3, 1923 TO JULY 24, 1926.

Numbers give dates of outbursts and quantities of coal and rock pushed out, *e.g.*, 15.5.24, 526/89 = 526 cars of coal = 263 tons + 89 cars of rock = 62.3 tons (1 car of coal = 500 kg. = ½ ton; 1 car of rock = 700 kg. = 0.7 ton.)

Arrows show direction of advance of heading.

several beds of coal of one mine; the average amount given off normally into the air currents in mining that may be counted on is about 26 cu. m. (918 cu. ft.) CO_2 per ton (metric 2206 lb.) of coal. In one case—in the Roschenvein of the Ruben mine—42 cu. m. (1483 cu. ft.) CO_2 per ton of coal was given off. In the upper mine levels the amount of carbon dioxide in the coal veins was generally less; the danger of instantaneous outbursts increases with depth while mining is going on.

The usual manner of gradual release of CO_2 gas in mining coal was known in Lower Silesia many years ago, but the first instantaneous outburst happened in the Sophie mine in Lehmwasser in 1894 and this accident was kept a secret for many years. Not until 12 years later were

there occurrences of instantaneous outbursts of carbon dioxide in other mines of Lower Silesia.

In the 10-year period from 1890–1900 in Lower Silesian mines, one outburst took place; from 1901 to 1910 there were 12; from 1911 to 1920, 242, and from 1921 to 1930, more than 320 outbursts.

Of the 573 outbursts enumerated previously, 400 outbursts happened in the Ruben mine (Figs. 3 to 6). By all the outbursts that happened from 1894 to 1925 (the dates of the outbursts from the latter year to the present are not known exactly) a total of 440,037 tons of coal and rock was thrown out; and in the Ruben mine alone 359,527 tons. In the Wenceslaus mine up to 1925 only nine outbursts had occurred, which threw out 418 tons (metric) of coal and rocks. On an average for each

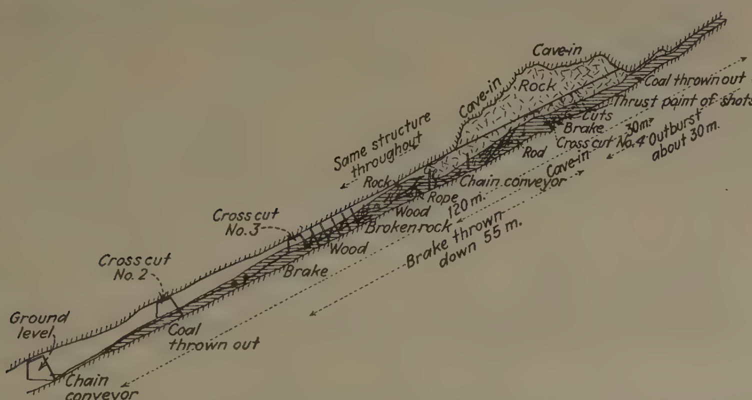

FIG. 5.—CROSS-SECTION OF ANTON SEAM IN WHICH AN OUTBURST OF CO_2 OCCURRED MARCH 13, 1926.

outburst, the tonnage burst out was about 100 tons, in the Ruben mine alone 112 tons, and in the Wenceslaus mine 46 tons. One single outburst in the Ruben mine, which occurred in February, 1925, threw out 1310 tons of coal.

As yet an exact way to anticipate outbursts is not known. As a general rule it may be said that any natural change in the working condition must be a cause of suspicion; if the coal face has been giving off gas regularly and the flow discontinues suddenly, this is a cause for apprehension. On the other hand, it is equally alarming when a coal face which has not been giving off gas begins to give it off suddenly. Furthermore, a change of soft coal to hard coal or vice versa will cause suspicion that there is a cluster of carbon dioxide storage places liable to outburst instantaneously.

The method of boring exploratory drill holes has been tried, to determine the degree of danger by determining the presence and pressure of the gas in the boreholes. As a general rule, it may be said that the researches in tests of this character have as yet reached no exact con-

clusions. It has been found that instantaneous outbursts may not take place in a coal face, although boreholes in it may find gas that contains 98 per cent. CO_2 and continue to give off this gas regularly. Also, the pressure of the gas in the coal seams under usual conditions is as a rule very low. Pressures as determined in boreholes of only 1.5 atm. abs. (about 7 lb. per sq. in. above atmospheric) are found either in dangerous beds or nongaseous seams, so that gas pressure as determined in boreholes is not as a rule an index of the degree of danger from outbursts. Geophysical and other methods so far tried have proved ineffectual in

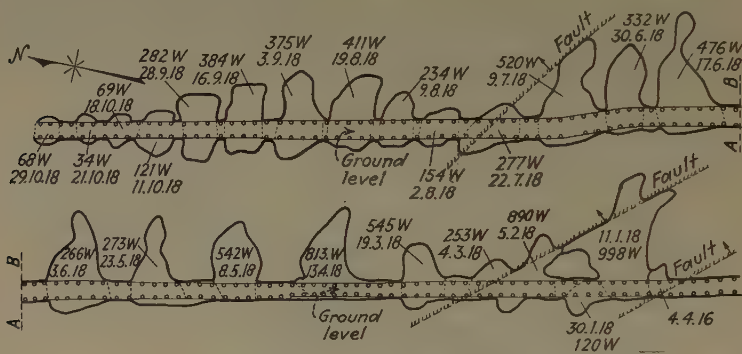

FIG. 6.—EXAMPLES OF OUTBURSTS OF CO_2 IN LEVEL GALLERY OF FERDINAND SEAM, RUBEN COLLIERY.

Area enclosed in heavy lines shows extent of each outburst. Figures indicate carloads (W) of material removed and dates of outbursts.

predicting an outburst of carbon dioxide. As a matter of interest, it may be remarked that trials have been made by men with divining rods, but have been of no value.

It is the rule before advancing a working place to drill two boreholes of 2 m. (6.6 ft.) in different directions. These holes may determine whether the coal has changed in character, whether there are indications of a fault or other valuable information. Moreover, it may be observed whether the carbon dioxide in the seam is under a high pressure and whether the gas issues in a regular flow. In a bed containing a certain amount of carbon dioxide, which issues regularly, the gas may be felt underneath the boreholes like a cold stream of water running down out of the hole; also, the smell of gas is noticeable.[2] Further, its presence may be demonstrated by the extinguishing of the flame of a safety lamp.

Thus exploratory boreholes have proved a valuable means for determining the condition of the seams and the degree of their saturation with carbon dioxide but they are not an absolute means for detecting the imminence of a threatening outburst.

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[2] When pure, CO_2 is odorless. Doubtless there is some other gas or substance present to give the odor mentioned. The effect of flow from a borehole was also observed by the undersigned during a visit to a "face" in 1928.—G. S. Rice.

Mining and Safety Measures in CO_2 Mines

When mining is begun in coal beds in which carbon dioxide is found, it is necessary first of all to search in a safe manner for gaseous areas. For this purpose, in developing an unworked area of a mine, a number of inclined passages and boards or sublevels are driven for fixed distances. The distances for the boards or passages are 30 to 60 m.; for the inclined passages, 100 to 200 m. All these passages are driven by blasting with strong shots (shock blasting—*tir d'ébranlement*). When no instantaneous outbursts of carbon dioxide are caused by this blasting, it may be supposed that no outburst clusters are in this section of a bed and the extraction at the face may be done by the usual drilling, small blasts and pneumatic hammer work. Besides the exploring of the district, a network of passages driven in this way will be useful by causing the veins to give out gas; in a similar way, in the district of Gard (France), *quadrillage* is done.

When outbursts of CO_2 have been caused by ordinary blasting, all the extraction thereafter in the district must be done by shock blasting; any work by hammers, picks and hoes is strictly forbidden, in order to avoid any unexpected shocks.

In Lower Silesia great reliance is placed on blasting in outburst areas, as it is known that imminent instantaneous outbursts of carbon dioxide may be brought on by strong blasts. So all shooting is done at predetermined times—and at these times the whole crew of the mine is brought out to places of safety, so that human lives may not be risked. All the shots are ignited electrically from safety stations after the miners have retired to these shot-firing stations. These stations are protected by heavy double doors of iron, made tight with felt (Fig. 7). By shutting these doors the entrance to a panel is closed hermetically. Tubes pass through the frames at one side of the doors. There is a sighting tube with thick glass windows, through which a safety lamp put on the ground before the doors are closed may be observed; also, there is a smelling tube by which the carbon dioxide blown out to the station may be detected by smell.

Twice in a shift the blast holes are charged. The crew then retires behind the safety lock and the roll of miners in the station is called twice. Then the safety doors are closed, the separate sections inform one another and the central above ground by telephone that all the men are out of their respective areas. Then the order for shot-firing is given. The men wait ¼ hr. after shot-firing, then look through the observation tube to see whether the safety lamp is still burning, and also try the smelling tube. If CO_2 appears the doors are kept closed and no one is allowed to go through the doors into the area until the carbon dioxide has disappeared.

After ¼ hr., if no CO_2 has been indicated the area is inspected by the air men or safety men, and not until the safety men have advised

that an area is free from CO_2 is the crew allowed to go back to the working places. The loss of working time in CO_2 mines caused by following of these precautions is great. It is counted that on an average in the CO_2 sections there is a net working time of not more than 5½ hr. out of the regular work time of 8 hours.

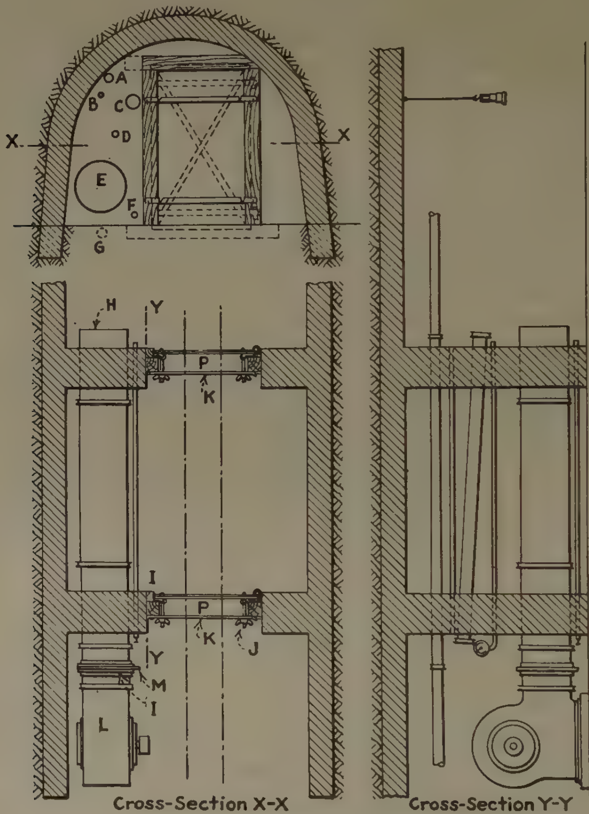

Fig. 7.—Plan and cross-sections of air lock.

At suitable places in the gallery at least two tight doors are built by means of solid rock or concrete walls. Electrical shot-firing stations outbye this air lock.

A. Compressed-air pipe.
B. Shot-firing cable.
C. Inspection pipe.
D. Air-pressure indicator.
E. Emergency ventilating pipe.
F. Testing pipe.
G. Drain.
H. Inbye to panel working.
I. Felt lining.
J. Tightening screws.
K. Supporting beam.
L. Emergency fan and pipe.
M. Damper.
P. Doors.

Besides the heading and winning by shock blasting, several other precautions ought to be used in mines subject to outbursts of CO_2. Backfilling of excavations with dead rock must be done most carefully, in order to support or cushion the descending roof stratum and to avoid

ruptures or shocks. Backfilling used to be done by hand and partly (Ruben mine) by hydraulic stowing. It was proposed to drop this method and work without gobbing or back filling or at least to do only light "walls," "cogs" and "pig-stys" (rock-filled cribs), because it was supposed that shocks would better be avoided in that way. In mines with so many faults as there are in Lower Silesia, this method is not needed to avoid bumps and for other reasons cannot be recommended.

The ventilation system is very important. Air is conducted now in a reverse manner to that employed in mines having fire damp; that is to say, which employ the ascensional method for return air. In Lower Silesia fresh air is conducted along the top level to the workings and down over them to the lower level and from there to the air shaft. This manner has proved useful, as CO_2 is heavier than air and tends to sink to the deepest point of the mine.

Ventilating fans ought to be of a capacity and strength to be greatly speeded up, so that CO_2 will be carried out of the mine in a short time. It is customary to drive the fans by electric motors of ample capacity.

The pipe lines of local blowers are placed along the floor, and as ventilators are operated exhausting, carbon dioxide given off at the face is more readily carried away.

The quantity of fresh air amounts on an average in Lower Silesian mines to 10 to 12 cu. m. (350 to 390 cu. ft.) per minute for each man of the shift underground. The percentage of CO_2 in the return air averages about 0.45 per cent. (including the relatively small amount of gas of respiration).

Lighting underground is done by storage-battery electric mine lamps carried by hand and by stationary incandescent electric lights, but in any work place at least one flame safety lamp must be kept burning in order to indicate when CO_2 is being given off in dangerous amounts.

By these safety measures, which have been introduced in the course of years of experience, it has been possible to limit the accidents caused by carbon dioxide outbursts. Nevertheless many have occurred, on account of the magnitude and the frequency of the outbursts.

From 1891 to 1930, about 573 instantaneous outbursts of carbon dioxide occurred in nine mines (four of them are now shut down). As a result of the 573 outbursts, 23 accidents happened, by which 221 miners were killed—151 of them on account of the great outbursts at the Wenceslaus mine on July 9, 1930. From 1921 to 1929, inclusive, 289 instantaneous outbursts of CO_2 took place, but accidents happened in only 7 cases. As several of these accidents were caused because miners had not observed the rules given to them, it may be said that the safety measures described are well fitted to prevent accidents.

Instantaneous Outburst at Wenceslaus Mine, July 9, 1930

The recent outburst catastrophe in the Wenceslaus mine was one of the largest instantaneous outbursts of carbon dioxide that has occurred in Lower Silesia, and it resulted in the saddest catastrophe that ever happened in that coal field. It occurred on July 9, 1930, in the afternoon, in the mine workings of the Curt shaft of the Wenceslaus mine at Hausdorf, near Neurode. One hundred and fifty-one miners were instantly killed. Outbursts of CO_2 in this mine had been rather rare until this one. The existence of CO_2 gas in the coal beds of this mine was unknown before 1915, and the appearance of it is still limited to the eastern part of the coal area mined.

The outburst happened in the seventeenth section in workings in the extreme east of the mine, in the Wenceslaus coal bed. The Wenceslaus bed is the highest seam of the several beds of coal and none of the other seams underneath have been worked. (Fig. 8.)

This bed of coal is about 1.80 m. (5 ft. 11 in.) thick; the coal is medium hard and there is a rock band in the seam. The roof and the floor are composed of sandstone. The coal contains carbon dioxide, which is given off in mining operations rather regularly. The pressure of gas in the boreholes had always been small.

The winning of the coal seam was done by longwall faces each about 190 m. (627 ft.) long in which a pack-wall system is used. The working where the outburst occurred was at a depth from 260 to 350 m. (855 to 1155 ft.) underground and extended from the third up to the second "sole" (sublevel). At the time of the outburst there were 55 men at the face and in the whole seventeenth section 85 men. The fresh air, 350 to 400 cu. m. (13,000 to 14,000 cu. ft.) per minute came from the third level and went up the longwall work to the second level and from there to the air shaft. "Holing" or cutting in the coal used to be done with a type of drilling apparatus until the spring of 1930. At that time in the upper (raise) third of the longwall face the gas came out in larger volumes and several outbursts occurred in the levels. So the use of the cutting apparatus was forbidden in this part, and was allowed only in the lower (dip) two-thirds of the face. The upper third from this time on was worked by blasting off the solid, and so also in the headings. Later on, in the upper part of the face several little faults were cut and one outburst took place in this part of the longwall.

The blasting was done when the crew had retired behind the safety doors. On July 9, blasting had been done at 1:15 p. m. At the beginning of the second shift, at about 2 p. m., no peculiar conditions were discovered. The miners began to work and drilling was done. Several minutes after 4 o'clock an immense instantaneous outburst of carbon dioxide occurred. In a moment the gas filled the whole seventeenth section and after one or two seconds had traveled to the main haulageway of the third

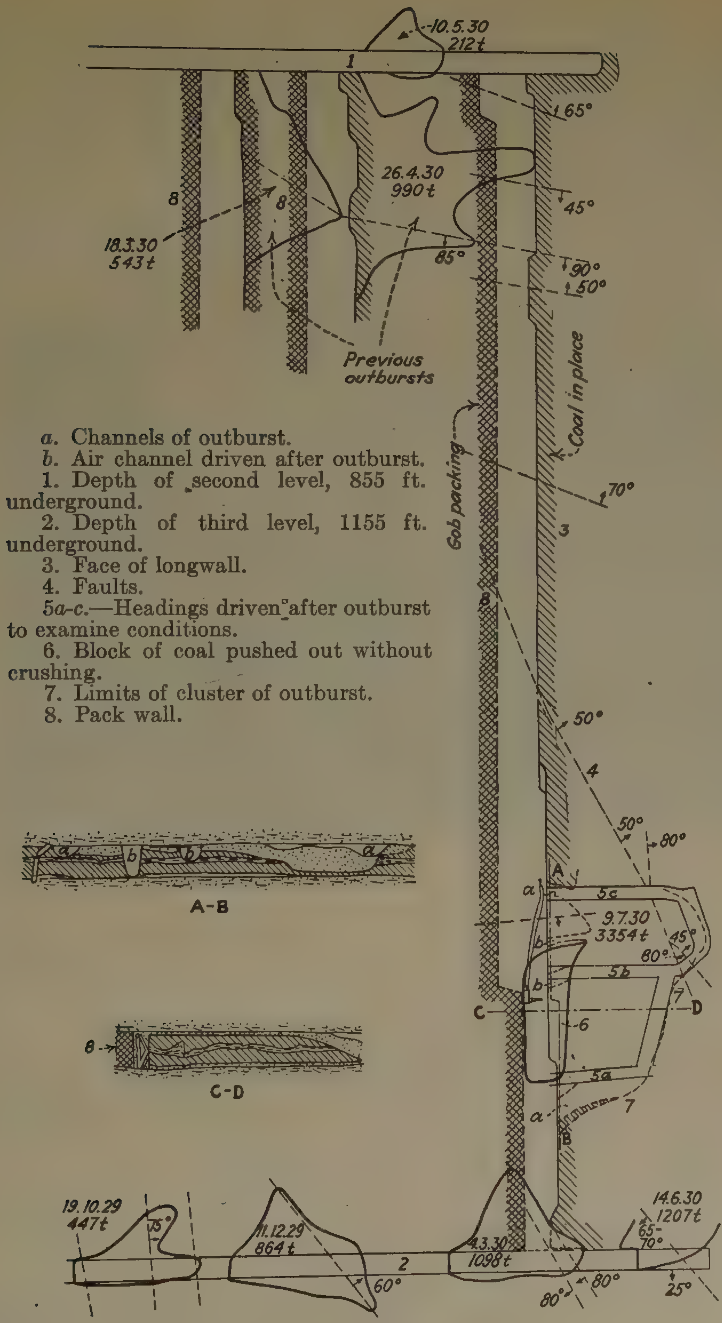

FIG. 8.—MAP AND SECTIONS OF THE LONGWALL AT WENCESLAUS MINE, WHERE THE OUTBURST OCCURRED ON JULY 9, 1930.

Former outbursts are marked by dates and quantities blown out are given in metric tons.

Wm. R. Chedsey.

level, about 750 m. (2475 ft.) from the place of the outburst. There the gas first ran against the main fresh-air current and so was carried into the neighboring section, panel 18, which it filled, both the gates or headings and the face. All men in this and the next panel being killed, several minutes passed before occurrence of the outburst became known on the surface.

Rescue and Recovery Work

The rescue men of the Wenceslaus mine succeeded first in closing the safety doors of section 17, so that the CO_2 could no longer issue from that

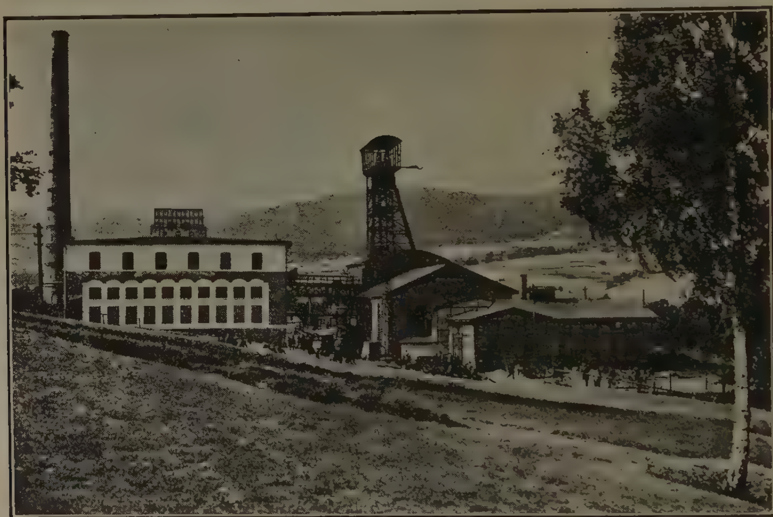

Fig. 9.—Shaft buildings at Wenceslaus colliery, showing topography of country and mountains of the Eulengebirge.

panel. The fresh-air current was then powerful enough to gradually carry away the CO_2 in other parts of the mine, especially section 18. Immediately after closing the doors of 17, the rescue men of Wenceslaus mine and the neighborhood mines (Ruben, Rudolph and Sophie), who had arrived meanwhile, advanced to section 18 and were able to save 49 men. The rest of the shift of the mine unfortunately were dead. This rescue work was done without respiratory apparatus, which might have hindered their progress through the narrow gates or passages.

Although ventilating fans were forced to their maximum capacity, the carbon dioxide retreated very slowly from section 17; probably more gas continued to issue out of the center of the instantaneous outburst area for a long while. So it was impossible to examine this section or panel before the afternoon of the following day. Dreadful was the view when the safety doors were opened.

As concerns the physical effects of the outburst in section 17 in the lowest third of the longwall face, a great block of coal, 16 m. (53 ft.)

long by 8 m. (26 ft.) deep, or wide, had been pushed out of the wall a distance of about 2 to 3 m. (6 to 10 ft.) without crushing. Moreover, an immense amount of very fine coal dust had been blown out and filled the longwall face working place for a length of 130 m. (429 ft.) from floor to roof. About 3000 cu. m. (3924 cu. yd.) of coal and rocks had been blown out and approximately 30,000 cu. m. (1,059,400 cu. ft.) of carbon dioxide had issued. The surface in the plane of the bed of the outburst area was about 700 sq. m. (837 sq. yd.)

After the longwall face had been cleaned up, three headings were driven in the place where the outburst had happened, in order to examine the physical conditions. It was found that a fault had existed about 15 m. (50 ft.) ahead of the face immediately prior to the outburst, which had been unknown. It is conjecture that a great cluster of carbon dioxide had been stored in or adjacent to that fault. The cluster, however, could not be detected in advance, the holes bored to examine the wall being not deep enough. Just at the moment when the wall of coal retaining this accumulation of CO_2 was weakened by advance of the face, the pressure of the gas in the cluster became strong enough to burst out suddenly without warning at one small place and then the torrent of gas pushed away the block of coal mentioned and threw out the immense amount of extremely fine coal dust. This explanation, however, is only a personal opinion. The government of the mine believes—and so do the officials—that the outburst was caused by rupture or by a shock imparted by the strata. The precise reason why the outburst could happen at all and how it caused the observed effect will remain a sad secret, all eye-witnesses being dead. As yet all comments and opinions reached by men of science and experience have not cleared up satisfactorily this unexpected and unanticipated disaster.

Mine work in that part of the mine has been stopped since that time and Wenceslaus mine was indefinitely closed on Jan. 28, 1931 because of the high cost of coal production. This conclusion is a hard blow for the whole district; about 3000 men have lost their opportunity for work, because there are no means to provide other work in this region.

The commercial situation may be understood when the conditions surrounding our Lower Silesian mines have been explained better. On account of safety measures necessary to prevent outbursts, the net daily working time of the miners is rather short (about 5½ hr.) and so the average coal production of a man is rather small. The average daily production per miner at the face is about 2000 kg. (2.20 tons of 2000 lb.); the average daily production per man underground is about 1000 kg. (1.10 tons), and the average daily production per man, surface and underground, is about 750 kg. (0.83 tons).

Although the wages are rather small—the pay of a miner at the face is about RM 6 ($1.44) per day—the proportion of labor cost is too great

compared with the selling price of the coal. Furthermore, there is needed per ton of coal produced 0.045 cu. m. (1.59 cu. ft.—19 board measure feet) of wood and about 200 g. (0.44 lb.) of explosives per ton of coal hoisted. Also, contributions to labor insurance, taxes, etc. are all very high compared with similar contributions in prewar years.

The selling price of the coal per ton at the mines is not more than about 9.20 M ($2.21 per metric ton or $2.00 per ton of 2000 lb.) on an average. The resulting situation may be interpreted by the sad conditions of the Lower Silesian mining district. The coal is generally soft and suited more for industrial works than for household purposes. The number of manufacturing plants in the east of Germany being limited, the greatest part of the production was sold formerly to the neighboring Bohemian districts and to Austria. After the peace of Versailles, the exportation of coal to Czechoslovakia became rather difficult; the exporting to Austria has stopped altogether. So the conditions of the mines in Lower Silesia have become more and more difficult, on account of the mining hazards as well as on account of the market curtailments.

DISCUSSION

(*Albert O. Hayes presiding*)

[This discussion refers also to the paper by G. S. Rice, beginning on p. 75.]

Bumps and Outbursts at Crow's Nest Pass

B. Caufield, Coal Creek, B. C. (written discussion).—A severe bump occurred on the main level (Fig. 10) in our No. 3 mine at 9 p.m. on Jan. 6 of this year, when only the fire boss was in the mine. The entry previous to the bump was about 12 ft. wide, 9 ft. high, and closely timbered, as the ground was faulty. The bump completely closed the entry, threw up the floor, crushed the ribs and caved the roof, rock from 3 to 5 ft. thick falling for a distance of 250 ft. We could clean only through the damaged entry from the outbye side, and because tracks, ties and timber were mixed with caved rock, heaved floor and crushed sides, it was almost as bad to get through as driving in the solid rock. The mine was idle from January 6 to 29. The parallel entry, only 50 ft. on the lower side, was not damaged, but the airway 150 ft. on the upper side was crushed tight. Notice the size of pillars on the upper and lower sides of the entry; also the faults as marked on the plan. *The most remarkable feature about this occurrence was that not one single timber was broken with the bump.* All the legs on the upper side of the damaged part of the entry were "kicked" out, letting the timbers fall and the roof cave.

The workings (200 ft. higher) in the upper seam No. 1 East immediately overhead showed no sign of the occurrence of a bump.

The No. 3 seam is about 6 ft. thick, with 2 to 3 ft. of medium hard shale in the floor, and under this is a seam of inferior coal 2 to 3 ft. thick. It is very gassy, and the gas exudes freely, often extinguishing a safety lamp held in the hand. When a bump occurs, and this lower coal is burst up violently, large quantities of gas are given off, in a manner similar to an outburst. This gave rise to the idea in early days that gas pressure was the cause of bumps.

Another incident, an outburst, happened on January 7. We stopped all work in the outburst area of this mine after the fatal accident in 1928. We are now working a few places on the fringe of this area with a view of experimenting in pillar drawing, and three places are going in one of the big blocks of coal, splitting it into smaller pillars. In these places we keep advance drill holes of not less than 15 ft. long ahead of the face, but are in the first drilling, bored from 25 to 30 ft. Close to these we had a place started in another large pillar, driven up about 20 ft. which was stopped on March 6, 1930. The drill hole was in 19 ft. on that date.

Owing to an accident two miners were sent into this place on March 6 of this year after it had stood exactly 12 months. They measured the borehole, found it intact and 19 ft. long. On the first shift, the miners loaded 14 tons of coal. On the second

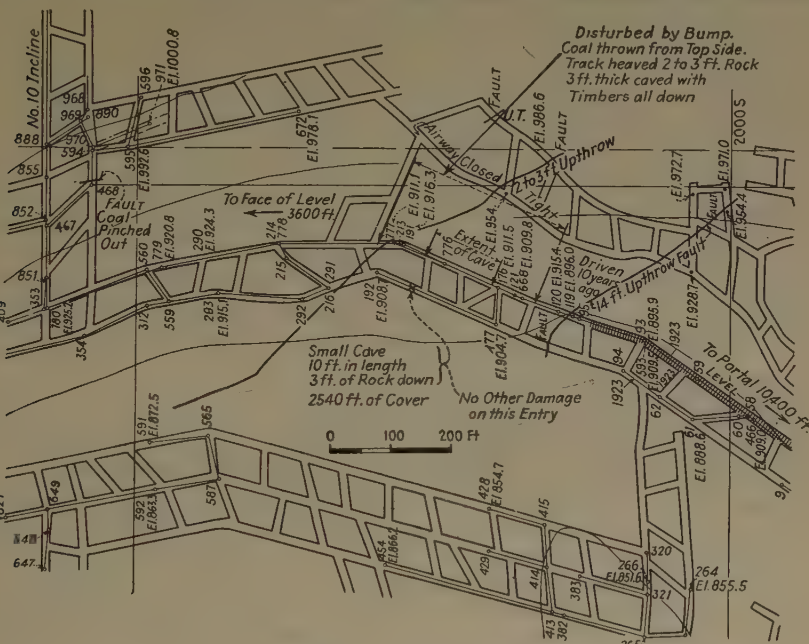

FIG. 10.—MINE AT CROW'S NEST PASS.

morning, shortly after the miners commenced work, the place blew out, discharging 60 tons of coal and a large quantity of methane. Here again, as in several other incidents, the borehole gave no indication of an impending outburst. A short record of outbursts after holes had been drilled is appended (p. 105).

Ground Stresses as Cause of Outbursts

These incidents, added to several others, seem to me to support the theory that ground stresses set up by former geological movements are the primary causes of both outbursts and bumps.

In the regular working of our No. 1 East mine there is a constant steady roof movement over large areas, roof creaking, thudding, bumping and trembling, right from the coal face upwards for some distance. Back on the roadways there is a general uneasiness in all the surrounding strata. When this condition is normal and the general all-round movement steady, everyone underground is contented, but when it stops and quietness prevails, there is an apprehension of danger and everyone is afraid.

It seems as though the movement has been arrested and when relieved something serious will happen. One really must experience this condition to understand it. Long ago I concluded that this movement is caused by adjustment of stresses in the strata after some of the coal has been removed, and when there is an exceptional stress caused by former geological movement, which could not get free at the time it was formed, it will in time, when the resistance is removed, burst along the line of least resistance (the open spaces in the mine) and cause outbursts or bumps. If the coal is subjected, and there is reason to believe it is, to the geological movement that caused these stresses, the grinding of the coal would set free gas which would fill the porous mass, loosen as it were the absorptive and adsorptive combination qualities and leave a highly charged coal and gas condition in combination with an extreme geologic latent force, which relieves itself by outbursts, when given a chance, and sets free large quantities of pent-up gases.

Mr. Rice, in referring to bumps in Coal Creek,[1] says: "There was surprisingly little gas given off after the greatest bump occurred," etc. This may or may not be true. The bump occurred on the main intake airway and if it gave off gas the gas would be carried immediately into the return airway. Owing to the confusion, and the fact that a man was buried under a cave at the time, no one entered the return airway. In the big bump in No. 2 mine (in 1908), 900 ft. of the main entry caved in, a larger area was damaged than in No. 1 East mine, and 24 men were imprisoned inside the bump, four of whom lost their lives. The only way into the mine was through the return airway and the gas prevented us from getting in for 7 hr. It was estimated at the time that over 3,000,000 cu. ft. of gas was given off. I have known of other bumps that gave off large quantities of gas.

Precautions against Outbursts

I notice that the outburst in the Wenceslaus mine on July 9, 1930, in which 151 lives were lost, came off a longwall face about 627 ft. long and with a cover of only 855 to 1155 ft. The pushing out from the longwall face a block of coal 53 ft. long by 26 ft. deep, and I presume the thickness of the seam, 5 ft. 11 in., equal to approximately 300 tons, makes one wonder just what force could be present. This seems to upset the theory that the cause is gas pressure. I should like to know from what part of the face the very fine coal dust came—from the front of the face or from the back of the block of coal that was forced out. In one of our outbursts a large block of coal was broken off the solid, which contained 42 tons in one piece, but there was no fine dust in the outburst. I have described this before.

I notice that after cleaning up the outburst they drove three headings a distance of about 50 ft. and met a fault. This seems to me significant, as it shows that there had been prior geologic movement near by, thus supporting the theory of built-up ground stresses.

As to Mr. Wilson's statement (p. 101), "Just at the moment when the wall of coal retaining this accumulation of CO_2 was weakened by the advance of face, the pressure of the gas in the cluster was strong enough to burst out suddenly without warning *at one small place* and then the torrent of gas blew away the block of coal mentioned, etc.," I cannot see how the action as described would move such a large block of coal. If this enormous pressure came from the gas alone, as soon as it was given vent or broke out at one place it would discharge with diminishing pressure until exhausted, would it not? Like tapping a compressed-air receiver or line. I would opine that if it were gas pressure alone it would move the block of coal first, then some part would give way and allow the pent-up gas pressure relief. However, I am inclined to think there were ground stresses present, and the fault being so near would tend to support this view.

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[1] See page 77.

I agree in the main with Mr. Rice's summary of the theory of outbursts. I cannot, after our experience, agree that boreholes are of any assistance either in warning of the danger or in informing one of the likelihood of an outburst.

I do think that blasting of the coal face in outburst areas should be tried extensively, using Cardox in seams where the outburst gas is methane, and allowing a definite comparatively long interval, say a shift, to elapse between the time of firing and the resumption of work.

I read Mr. Wilson's paper with keen interest. He mentions several of the characteristics in seams subject to outbursts that are experienced here, such as hard or tough coal preceding an outburst and change in the natural conditions of working giving rise to suspicion. On page 93, paragraph 3, he says, "A change from soft coal to hard coal or vice versa will cause suspicion that there is a *cluster* of carbon dioxide storage places liable to outburst instantly." I wonder just what he means by "a cluster" of carbon dioxide storage places.

He, too, had no success with boreholes, at least, as a warning of an outburst condition, although their holes are short compared to ours.

I was impressed greatly with the precautions taken in the blasting operations, but I consider ¼ hr. too short a time between firing time and returning to the face. I should favor a lapse of about 8 hr.—fire at the end of a shift and let a shift elapse before resuming operations.

Some Records of Outbursts at Coal Creek in Places Previously Drilled

On Oct. 21, 1922, there was a serious outburst from the face of 13 East entry, No. 1 East mine. Mr. Stubbs, who was then Safety Inspector, reports as follows:

"This place was drilled together with the crosscut from 14 East on the morning of Oct. 21 (date of blow-out). The crosscut had an 18-ft. long hole, while the face of 13 East had two holes drilled. When cleaning up it was found that the hole on the right side had passed through the rash and into the top coal immediately in front of and 1 ft. from back of the cavity broken into this softer material. It should also be noted that this place was bored on Oct. 16; was idle on Oct. 19 and 20; was bored on the morning of Oct. 21 (two holes), no gas reported, but blew out on the afternoon shift of Oct. 21 at 9.30 p. m."

Outburst on Nov. 5, 1926, in No. 2 crosscut, 17 room, 16 East slope, No. 1 East mine. A drill hole had been drilled in this place 17 ft. long, 3½ hr. before the outburst, and the place had been idle from Wednesday at 2 p. m. until Friday at 8 a. m. Outburst occurred at 9:30 a. m.

Outburst on Jan. 25, 1928, in 20 room, 16 slope. Place drilled on Jan. 20 and 21, 18-ft. hole each shift. Twelve feet of the second borehole had been worked off, leaving 6 ft. still on at time of outburst.

Outburst March 2, 1928, in 9 East entry. Place drilled Feb. 23, 18-ft. hole. Again drilled on Feb. 28, 15-ft. hole. Place idle on Feb. 29. Worked on March 1. Blew out on March 2.

Outburst April 4, 1928, in 20 room, 16 East slope. Place drilled March 27, 18-ft. hole. Again drilled April 4, 18-ft. hole. Blew out same date as drilling at 1:30 p. m.

Outburst April 5, 1928, in 9 East entry. Place drilled March 30, 17-ft. hole. Place drilled March 31, 17-ft. hole. Place drilled April 5, 18-ft. hole, 10 hr. prior to outburst.

Outburst June 8 in first right room off first left entry, 14 East slope. Place drilled June 4, 18-ft. hole. This place had worked off 10 ft. of this hole, leaving 8-ft. of hole at time of outburst.

Outburst July 9 in crosscut off 20 room, 16 East slope. Place drilled July 6, 18-ft. hole. This place had only worked one shift from time hole was drilled until shift on which outburst occurred. There was 15 ft. of hole on at the time of blow-out:

Outburst on Aug. 30, 1928 (fatal) in No. 1 room, off 20 room, 16 East slope. Place drilled on Aug. 23, 18-ft. hole. A cave was reported at the face of this place on Aug. 29, and it was not cleaned up and timbered when the driller visited the place on the morning of Aug. 30, so that he could not drill then, but he drilled the face of No. 2 off 20 instead, to a depth of 18 feet.

Outburst on Sept. 8, 1928, in 10 East entry. Place drilled Sept. 7, 19-ft. hole, the day previous to outburst.

Blow-out occurred on Nov. 29, 1928, at 10:15 p. m. in 10 East entry. The place was drilled on the 19th to a depth of 27 ft., so that about 19 ft. of hole was still on at time of outburst.

A severe bump occurred on April 8, 1931, at 8 p. m., when no one was in the mines. The bump shook all the buildings in the camp and reports were sent to the office by several householders. No sign of the bump was found in the mines, and work proceeded as usual the following day. Such bumps are not unusual.

Pulverized Coal as Factor in Outbursts

H. Briggs, Edinburgh, Scotland (written discussion).—The papers by Messrs. Rice and Wilson are valuable by way of emphasizing the results obtained in regard to the origin of outbursts of gas underground and to possible ways of coping with the danger. My own experiments (to which Mr. Rice refers) related only to the cause of outbursts, and it is proper that my remarks should principally refer to this side of the subject.

Mr. Rice has had exceptional opportunities of studying the outburst problem in many lands, and it is important to realize that, in the light of his experience, he has arrived at conclusions as to the cause of these large-scale discharges of gas coinciding in all essentials with those advanced by workers, such as Graham and myself, who approached the problem from a different angle. His paper provides an admirable summary of the findings of independent investigators, among whom he stands in the front rank, and whose views are unusually unanimous. Briefly, they are to the effect that coal itself is able under pressure to hold by adsorption all the gas, whether CO_2 or CH_4, that is needed to account for outbursts, and in this direction the problem may be considered as solved.

In giving the quotation, "The coal ejected by these outbursts from the solid is dry," from my 1921 paper, Mr. Rice misunderstands the expression "from the solid." It was not intended by that expression to convey the impression that the outburst coal was not disintegrated *in situ;* it was used to distinguish outbursts from coal from those taking place from the roof or pavement of the seam. In the paper in question, indeed, I strongly advocated the view that a necessary condition for these outbursts from coal was that the coal should be in a pulverized condition *in situ*, and in this respect, as in most others, my opinion coincides with Mr. Rice's.

I think there is little doubt that the pulverized coal would be compelled to discharge its gas if it could be wetted, since it appears to have a greater affinity for water than for fire damp or even for carbon dioxide. Mr. Roblings, whose name is well known in connection with the study of outbursts at Ponthenry colliery, attempted to make use of this inference by injecting water from the rising main of the colliery pumps through boreholes into the coal, but was unable to apply the method satisfactorily, owing to the facility with which the boreholes choked.

Mr. Greenwald's diffusion tests are decidedly interesting, and I hope that he can be induced to continue them. As Mr. Rice states, the indication that C_2H_6 has been replaced by CH_4, in consequence of its passage through powdered coal, is significant, and has implications considerably exceeding the boundaries of the specialized inquiry which Mr. Greenwald is undertaking.

I observe that Mr. Rice subscribes to the view current among some geologists that the carbon dioxide of the Gard and Lower Silesian districts originated in igneous rock. I have not had the opportunity of studying that problem on the spot, and therefore am not rash enough to doubt its possibility. It will be enough to observe that, in accordance with the views of those who have recently studied the evolution of coal, more particularly Fieldner, Hickling and myself,[3] the effusion of carbon dioxide appears to be a principal result of the slow development of coal, this gas being more particularly emitted during the change from lignite to bituminous coal. Succeeding stages in the evolutionary process involve the expulsion of greater proportions of CH_4. It seems to me, therefore, quite within the bounds of probability that the carbon dioxide of the Gard and Lower Silesian seams may merely be a product of their own development in rank.

In view of the limited success attained by the use of advance bores in coal as a means of indicating outburst material or as a means of draining that material, I doubt the efficacy of the second of Mr. Rice's precautionary measures. Experience has shown that if a hole enters soft coal it tends almost at once to be choked by the pulverized material and to give little or no indication in the shape of discharge of gas. If a system of boring, however, were introduced enabling the pressure on the drill rods to be registered, a fall in that pressure indicating that the bit had entered soft stuff might turn out to be a more useful indicator.

Warning Noises

Mr. Rice has referred[4] to the warnings of impending blow-outs at the Cassidy mine, which took the form of cracking and splintering of the coal at the face, often followed by a sound resembling a quick-fire gun or a compressed-air drill. I do not, however, find in these present papers any reference to the important question of warning noises. At Ponthenry there is a striking change in the nature of the noise produced ahead of the faces, owing to the cracking of the coal under roof weight, the noise being sharp and distinct in hard anthracite, but dull and muffled if an outburst area lies immediately in advance. This change is, indeed, at Ponthenry the chief indicator of the occurrence of a danger zone in front of the faces. It would be of much interest if Mr. Rice or Mr. Wilson were able to enlarge upon this aspect of the problem in their replies to the discussion and to inform us whether these audible indications are reliable.

Outbursts at Ponthenry Colliery

G. Roblings, Ystradgynlais, Swansea, South Wales (written discussion).—The writer is of opinion that in order to show how certain conclusions have been arrived at, a short reference to characteristics attending some outbursts and the results of what we called "volley firing" as practiced at Ponthenry colliery in the anthracite area of South Wales will be valuable.

The outbursts all took place in one seam—the lowest workable seam in the Gwendraeth Valley in the extreme west of the coal field; five other seams were also being worked, but no trouble of any kind had been experienced, although areas of very friable coal were met with from time to time. All the seams had been followed from their outcrops downwards at a gradient of approximately one in four and one-half. In the lowest seam, also, areas or belts irregular in form of friable coals had been met with.

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[3] H. Briggs: The Evolution of Coal. *Jnl.* Soc. Chem. Ind. (1931) **50,** 127.

[4] G. S. Rice: Discussion of Outbursts of Gas, and Methods of Working Seams of Coal Liable to Them, by G. Roblings. *Proc.* So. Wales Inst. Engrs. (1927) **43,** 35.

The seam consisted of two coals; the top coal 8 in. thick, consistently strong and hard, while the lower coal, normally 2 ft. 8 in., varied up to 6 ft. 0 in. thick, with a parting of 1½ to 2 in. of soft rashing. In this bed the soft coal was sometimes found. The roof overhead the top coal was strong clift or shale.

The seam had been worked down for about 900 yd. from the surface before there was any accident. The first real trouble occurred on Feb. 27, 1920, on a heading where the bottom coal had been followed in from just a few inches until in 22 yd. it had reached 2 ft. 6 in., the top coal having been consistent in thickness and hardness throughout.

The collier had hammered the coal with his sledge "to bring work into it"[5] and so caused considerable pouncing (or bumping) which increased in violence until the coal blew out. The men reached the slant, having run away on account of the heavy pouncing. A few men working in the main slant, a few yards below, who were obliged to pass the heading, described it as blowing "like a winnowing machine." These men succeeded in getting some distance above the heading before they collapsed into a stream of water which normally flowed down the slant to a pumping station, and which undoubtedly brought about their revival. All the lamps in the workings were oil flame and all were extinguished by the gas released. At 500 yd. up the slant, the description given by a man who was standing at that point was that it was like a gale whistling through trees. On subsequent examination, 46 yd. of roadway was found filled within 10 in. of the roof by very fine coal, and the surface had the appearance of having been wind-swept.

Exploration in this and subsequent outbursts has been taken in hand within 15 minutes without any breathing apparatus of any kind, and yet notwithstanding the rapidity with which it was done, some of the workmen (not rescuers) lost their lives. The writer is of the opinion that the very fine coal dust blown out with the gas current and carried for hundreds of yards with it, had more to do with the suffocation than the actual gas.

When the small coal was cleared away, a considerable area was found in which the coal formerly in place had been thrown out and the thickness of which had gone up to 13 ft. The whole of the belt of coal had been disturbed up to a pillar of strong coal of normal thickness which was unaffected.

A tram filled with coal and of a gross weight of 28 cwt., which had stood on the level road near the parting, had been blown out and up a gradient of 7 in. per yard (20 per cent.) and then thrown over on its side against timbers.

Sometime after this, it was necessary to drive the main slant through this belt of friable coal, which was 6 ft. thick, and the road driven 10 ft. wide, it was decided to put in boreholes with the view to releasing the gas. Thirteen holes, each 3 in. dia., were put in to a depth of 9 ft. Considerable difficulty was experienced in keeping the holes open, owing to the very friable coal. Gas was discharged from the holes while boring proceeded. A sample was taken from one and analyzed by C. A. Seyler, Swansea, who found it to be 98.8 per cent. methane with just a trace of CO_2 and N. The discharge of gas ceased as soon as boring stopped, and it was subsequently found that the only gas released was from the coal which had been released by the drills, the coal immediately surrounding the borehole being as fully charged as prior to the boring. The coal was subsequently worked by merely scratching with the pick.

When boring in the coal to obtain the sample that was analyzed (which was being done by hand and twist drill) the gas was being released at such a pressure that the dust was blown back about a yard from the face of the coal and this was undoubtedly

[5] To shake the coal so that internal pressure stresses would be started to work and throw off coal on the free face.

in an outburst area while the portion in which the pressure was taken was in the near vicinity, and where a slight outburst afterwards took place.

The experience thus obtained convinced the writer that boreholes were not effective in draining the gases from the coal and showed that the gas could only be released by releasing coal as well.

It has to be remembered that all the areas or patches of friable coal were not subject to outbursts, but nothing had been found which would lead one to form an opinion as to which area is heavily charged with gas, hence every patch had to be considered under suspicion.

The large volumes of gas released with each outburst cleared away in a few hours, and it is a surprising fact that the normal discharge of gas from the coal was very small; in fact, less than that from other seams not subject to outbursts. This is quite consistent with the experience obtained when boring in the soft coal.

This differs from the experience in the Gard and Belgium where considerable quantities of methane are released from the working face.

The highest pressure obtained in a borehole in coal of a fair hardness was 7 lb. per sq. in., which is not a high pressure in strong coal, but with friable coals may be sufficient to maintain an unstable equilibrium requiring but a shock of sufficient intensity to entirely destroy the equilibrium.

In order to obtain some idea of the pressures exerted by the gas current, an experiment was made with a tram similar to the one previously referred to and on the same spot, by putting a spring balance between the rope and the tram, which was then drawn up slowly, when a pull of 616 lb. was registered. The rate at which the original tram was blown up is not known, but could not have been slower than in the experiment, hence the pull of 616 lb. could be taken as a minimum. This resolves itself to a pressure of 88 lb. per sq. ft. on the end of the tram. A further test was made to find the pressure required to tumble a tram, and the minimum was found to be 120 lb. per square foot.

Now 7 lb. per sq. in. is 1008 lb. per sq. ft.; it is therefore obvious what velocities can be obtained by escaping gases in confined spaces such as the roadways of mines.

One outburst was caused by the roof squeezing heavily on the soft coal. The fireman who happened to be in the working place at the time heard the roof working, and along with the collier stood behind the tram, watching the bed of clift forming the roof settling upon the coal and parting from the bed above. In a short time pouncing commenced in the coal and became so insistent that they considered it prudent to run away, and escaped to safety. A considerable volume of gas was released as well as very fine coal. The tram of coal, gross weight 30 cwt., which had been in the face had been forced back for several yards, and the back end forced up to the roof, which was 7 ft. high.

Another outburst flattened out completely some 12-in. sheet-iron air pipes. By a test on similar pipes it was found that it required a total weight of 4½ cwt. to produce a similar effect and a total weight of 504 lb. distributed over an area of 6 sq. ft. gave 84 lb. per sq. ft., a result closely approaching that found by the test on the tram.

Effects of Shock Blasting

After inspection of collieries in Belgium and the South of France, it was decided to adopt the method known in those places as *tir d'ébranlement*, and in a number of cases heavy outbursts were produced by this method. The writer will not take space to describe the method, but feels that he must describe some of the effects.

In two cases, telephones were fixed in close proximity to the places where the volleys were to be fired and were connected to the office phone where the firing was taking place, a distance of over 2000 yd. The firing was distinctly heard and in a short space of time other noises were heard particularly the sound of small coal being

thrown about, until the transmitter became covered with the coal. This volley had been prepared and fired adjacent to the site from which a very severe outburst had previously occurred with fatal results.

In the second instance of the successful use of the telephone, the time elapsing from the firing until the sound of coal being thrown about was found to have been 1 sec. and to have continued for 11 sec. In this case we reached the site in 15 min. and found that gas was flowing out in a strong current between the loose and the undisturbed coal, samples of which were taken. The analysis was as follows: CH_4, 88 per cent.; H_2, 2.8.

In all cases the coal has been very dry while the temperature appears to have been below the normal. A peculiar effect has also been that the small coal, in two or three days after being blown out, heated gradually to a maximum of 150° F., and then gradually cooled. After the outburst of June, 1928 I asked one of the workmen who was in the vicinity how he felt, and he replied that he felt the air much colder.

The whole of the blown-out coal was very fine and when freshly blown a person would sink to his knees without feeling any resistance.

The depth of shot holes comprising the volleys which were to be bored in the hard top coal was at the beginning not to be less than 6 ft., but after some experience it was ultimately decided to make 5 ft. the maximum depth. The reason which prompted this was that a volley had failed to blow out the front of the coal and on this being released, outbursts would occur. This was definitely proved by an outburst which had fatal consequences in July, 1928.

A volley of four shots had been fired at 11 p. m. the previous night but without any untoward results, but on the day in question what appeared to be a series of three outbursts followed each other in rapid succession, blew out from the upper side of the road immediately adjoining the face. Subsequent examination of the holes and the detonator leads, which was easily done because none of the actual face had been blown away, showed that the holes must have exceeded 6 ft. The explosive force of the four charges of 1 lb. each of Dynobel must have been dissipated in the soft coal. The writer is of the opinion that if these holes had been no deeper than 5 ft., the volley would have produced the outburst.

Crushed Coal Due to Earth Movements

The soft coal was found in belts with an orientation having a direction of south-southeast, this being the direction of a set of disturbances resulting from an earth movement which is responsible for the overthrusts so prevalent in the anthracite area, and, according to the late Sir Aubrey Strahan, is also responsible for the direction of the Neath and Swansea valleys.

In 1913 some outbursts occurred at a colliery in the Swansea Valley when roads were being driven across these disturbances which consists of a series of sharp folds. In this case the coal was also very fine.

It would appear therefore that this particular earth movement was responsible for the condition in which the coal has been found.

Examination of the friable coal *in situ* at Ponthenry, showed distinct evidence of crushing and had been intensely contorted. It is difficult to understand this in view of the fact that the top coal continued strong and hard with the normal cleavage only. The thin layer of rashings may have been the plane of horizontal movement.

There is evidence of considerable horizontal movement in many seams where there is a thin tough leather-like clay band about ½ in. thick between coal and roof resulting from the crushing of the softer shale by a sliding movement such as would be expected from a lateral movement of a thrusting character. One is tempted to say that the plane of separation between roof and coal often is the plane of horizontal faults, and crushing of the seam may be expected under some circumstances in combination with these.

The crushing undoubtedly puts the coal into a condition favorable for outbursts, but it is difficult to explain why all coal so crushed is not liable to blow out, or in other words, why all parts are not equally charged with gas.

A series of outbursts has in recent years occurred in the Yotsuyama colliery of the Mitsui Mining Co., of Japan, where there is a seam in two parts with lower bed about 7 to 8 ft. thick of fairly strong coal, overlain by a thinner bed about 8 in. A number of outbursts have taken place from this upper bed, resulting in the loss of a number of lives.

The coal here is strong enough to bore fairly long holes, and the gas cannot be held by the coal in the same manner as at Ponthenry, since it can be drained away by the boreholes. In several instances it was released at such a pressure that the boring tools could make little headway, but notwithstanding the success of these boreholes an outburst took place, suffocating two men. The explanation given was that possibly the boring tools had passed out of the gas-bearing bed. Four outbursts, however, followed ordinary blasting operations. Here again it is difficult to explain the different conditions in the two adjoining beds of coal.

In Belgium, where a seam known as the Cinqpaumes seam is liable to outbursts, there are two beds of coal about 12 in. each separated by a stone of varying thickness. The bottom coal is the stronger, in which the holes are bored for the shattering shots or *tir d'ébranlement.* The upper bed is very soft and liable to blow out. Mr. Rice says that the Belgian department does not permit blasting. I take it that this means ordinary blasting, as the practice referred to here has been permitted since about 1922.

Precautionary Measures

To refer to the proposals regarding precautions, the writer has mentioned the difficulty in keeping holes open in soft coal and the ineffectiveness of firing heavy charges too far from the front of the coal, but he is satisfied that the conditions in all seams liable to outbursts are not identical, since it was found that boreholes were of assistance in the Japanese mines in draining away the gas, and consequently measures to be adopted must be made to suit the local conditions.

A particular object which should be aimed at if practicably possible would be Mr. Rice's suggestion of separating the workings into panels, which could be isolated and each connected directly with the return airway.

Forms of Gases

The outbursts cited undoubtedly show that gases, principally fire damp, must be contained in the coal in two forms:

1. More or less unattached and found in the joints and interstices or at least so loosely held as to be released in working the coal.

2. That which is very closely attached to the coal and which can be retained by the coal for long periods in a very condensed form. The experiments described by Dr. Briggs thoroughly illustrate the point.

In the matter of what gases are released by these outbursts, it is interesting to note that when the writer was at Bessage in the South of France, Mr. Muscarte of the Bessage Concessions suggested that ethane was a constituent. The writer has not received any analyses of gases following an outburst which have contained ethane, and several of these analyses have been made by Mr. J. Ivon Graham. In a letter to the writer, Mr. Graham stated that he has "found by experiment that when freshly ground coal is placed in a dish and the evolved gases pumped off and analyzed, the composition of the gas first evolved corresponds to methane whereas the gases given off at a later period correspond more to ethane."

In view of the fact that an analysis[6] by J. Ivon Graham showed 88.8 per cent. CH_4 and 2.8 per cent. hydrogen of a sample obtained in about 15 min. after the outburst, there is no trace of any of the heavier hydrocarbon, and since these are released later, unless the disturbing effect of the firing of heavy charges of explosives has something to do with releasing them earlier, it does not appear that there were any of these in the Ponthenry coals.

R. D. Hall, New York, N. Y.—In previous literature it has been shown that these phenomena were more likely to occur in the deeper measures. It may be that the gases evolved, as has been suggested in Great Britain, are not gases in the coal but undiscovered polymers of those gases. It may also conceivably be true that the methane in wells is derived from polymers of methane and may give out heat in depolymerization, negativing in whole or in part, or even overcoming thereby, the cooling effect of physical expansion.

Causes of Gas in Coal Beds

G. Knox, Treforest, South Wales (written discussion).—Violent outbursts of hydrocarbon gases in coal mines are fortunately of rare occurrence and are usually confined to areas which have been subjected to great stresses due to earth movements subsequent to the formation of the rocks containing the coal seams. The outbursts at Ponthenry are good examples of this type, this mine being situated in an area of overthrust faults.

The ordinary outflows of gas into the workings of coal mines are the result of a process of degradation of the coal substance during the process of coalification. Large quantities of this gas would be forced under pressure into the fissures and, in the case of permeable strata, into the pores of the rocks. In shallow seams large quantities would escape to the surface, as in the Clyde Basin, where for years the escaping gas could be ignited at any time on the surface of the River Clyde near Bothwell Bridge. Similar phenomena were common at one time in the Rhondda Valley. Where the strata have been subjected to moderate stresses, pressures of 461 lb. per sq. in. have been registered,[7] but more frequently the outflow will continue for long periods at very low pressures. A good example of this type is the blower at Cymmer colliery, Rhondda, Glamorgan, which after 50 years of continuous discharge is still giving off 800 cu. ft. of almost pure methane per hour. This blower was struck in the shaft about 90 yd. above the 2 ft. 9 in. seam (after the shaft had been completed) when preparing a recess in the side of the shaft for a girder. Although the workings are now a long way from the shaft bottom, this blower shows no sign of giving out.

In some areas (particularly in synclines) large blowers of gas have been met in the fissured porous sandstones with only a normal discharge in the coal seams. In sinking Markham colliery (Monmouthshire) five such blowers were tapped (some of them preceded by a great inrush of water) in the sandstones above the chief workable coal seams. The two lower ones were the largest, each giving off about 1000 cu. ft. of gas per minute, the latter being preceded by a discharge of 12,000 gal. of water per hour.

E. M. Hann[8] describes an outburst from the floor of a coal seam at Aberdare with a fluctuating pressure varying from 34 to 8 lb. per sq. in., in which over 1,000,000 cu. ft. of gas was given off in 10 hours.

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[6] Analysis reported by G. Roblings: *Proc.* South Wales Inst. of Engrs. (1927) **42**, 48.

[7] L. Wood: Experiments Showing the Pressure of Gas in the Solid Coal. *Trans.* North of England Inst. Min. Engrs. (1881) **30**, 163.

[8] E. M. Hann: Notes on an Outburst of Gas at Aberaman Colliery. *Proc.* So. Wales Inst. Engrs. (1888–1889) **16**, 262.

The large amount of gas found in the workings of the South Wales coal field no doubt is due to the large quantity of gas stored up in the rock fissures, and the quantity stored up in the finely powdered coal between the cleats as a result of earth movement. The adsorbed gas contained in the powdered coal is liberated as soon as the coal is exposed to the atmosphere. The stored up gases in the fissures may be found issuing from subsidence fractures into the gob for a distance up to 100 yd. from the coal face; *i. e.*, up to the point where subsidence has more or less closed up the fractures and opened new ones inbye. This accounts for the fact that the first seam of a series worked in any mine gives off a larger proportion of gas than any of the others worked subsequently. It also explains why gas is seldom given off from roadways driven through the old gob which has subsided, although the gob—within a distance of 50 yd. from the working face—supplies a large proportion of the gas that pollutes the mine atmosphere.

Geologic Disintegration of Coal

J. S. Haldane, Oxford, England (written discussion).—I agree fully with the explanation given by Mr. Rice of sudden great outbursts of gas in coal mines. Mr. Wilson's paper fills various gaps in my own knowledge as to outbursts of carbon dioxide.

Previous to the publication of Mr. Graham's first paper (to which Mr. Rice refers) ideas as to the enormous volumes of methane which are often given off in a coal mine, as compared with the volume of coal extracted, were vague and unsatisfactory. This paper and the later one giving curves showing the volumes of a number of different gases taken up by coal at different pressures up to 34 atmospheres,[9] threw a clear light on the whole subject. In the discussion on the first paper, I pointed out that the data given for carbon dioxide explained the sudden great outbursts of carbon dioxide in the Gard district from patches of coal disintegrated by movements at faults, assuming that this coal had been saturated at considerable pressure by carbon dioxide from subterranean sources.[10] These outbursts had been known to me for 20 years, as well as sudden minor outbursts of methane in English collieries; and I had often puzzled over them before Ivon Graham's results were obtained.

Although an excellent account of the terrible accident at the Wenceslaus mine last July appeared in the *Times*, there was no account of the conditions which led to the accident; I therefore wrote a letter which appeared in the *Times* of July 14, to explain how carbon dioxide adsorbed by coal disintegrated at a fault explains such accidents. The following account of a tremendous outburst in one of the Gard mines about 35 years ago is quoted from that letter:

"The gas was liberated by the firing of a shot in sinking a shaft into a new thick seam. The precaution had wisely been taken of firing no shots except electrically from surface when the pit was clear of men. When the shot-firer at surface fired the shot he heard a rumbling sound from below, and noticed that air was blowing up the shaft, instead of passing down. Realizing what had happened, he called to the other men who were near, and they ran for their lives, except an engineman who, as he was in the engine-room, did not hear, and was suffocated. The gas and accompanying coal dust shot high into the air, and continued for a considerable time to discharge like a volcano. Being heavier than air, the gas came down, spread along the ground like water, and reached a village about a kilometer away. Here various fowls and dogs were suffocated, but fortunately the gas level at this distance was too low to kill human beings."

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[9] J. I. Graham: The Adsorption or Solution of Methane and Other Gases in Coal, Charcoal and Other Materials. *Trans.* Inst. Min. Engrs. (1922) **62,** 298.

[10] J. S. Haldane: Discussion of The Permeability of Coal to Air or Gas, by J. I. Graham, *Trans.* Inst. Min. Engrs. (1916–17) **52,** 353.

From the data given in Mr. Wilson's paper, it seems evident that the whole of the coal in a district subject to sudden outbursts of carbon dioxide is highly charged with this gas, though it can, of course, only come off slowly from solid coal. I should like to ask him whether he knows of any complete analyses of return air from one of these districts under normal working conditions? One would expect to find that the carbon dioxide added to this air would greatly exceed the oxygen removed from it by the usual oxidation processes, whereas in ordinary coal mines the opposite is, of course, true. In only one case known to me does much carbon dioxide come off along with methane from coal. This is in an Australian coal field; and the mixture is known as "bottom gas," because, though inflammable, it is heavier than air and flows along the bottom of a mine road.

I do not think that there is any good reason for concluding that the gas pressure in the patches of disintegrated coal which blow out is any higher than in the adjoining coal. The true gas pressures in coal are difficult to measure, however, except under especially favorable conditions in boreholes at the end of narrow headings in virgin coal. Cracks in the coal and overlying strata extend a long way in front of an advancing face, and so render measurements in ordinary boreholes impossible. Pressures of methane up to 32 atm. were measured by the late Sir Lindsay Wood in the north of England; and it seems probable that similar pressures of carbon dioxide may exist where outbursts of this gas are met with. The upper limit of gas pressure in a coal seam is probably set by the pressure of a column of water from the surface to the seam; and this perhaps explains why gas outbursts, when they occur, become more serious as the depth increases. One might expect that as a face approached a patch of softened coal, the gas adsorbed in this patch would blow out quietly through the cracks. But apparently the cracks are apt to become blocked in the manner suggested by Mr. Rice, by the lumps and fine dust passing into them, so that the gas is pent up till the face is close.

I agree entirely with Mr. Rice's inference that the coal blown out with the gas has been previously disintegrated by earth movements. Ivon Graham found that small lumps present in the coal from one of the Ponthenry outbursts in South Wales did not, even when they were saturated with carbon dioxide at 50 atm., form any dust when the pressure was released suddenly. Hence the rapid emission of gas does not seem to form the dust which accompanies the emission.

Higher Hydrocarbons in Coal

In connection with Mr. Rice's remarks as to higher hydrocarbons in coal, it seems to be a general rule with English coal containing gas that when the gas is exhausted from the coal into a vacuum the last fractions of gas contain a considerable proportion of higher hydrocarbons, although in samples of return air from the same seam the inflammable gas is very pure methane. This seems to be most simply explained by the fact that the higher hydrocarbons, with their higher critical temperatures, are much more firmly adsorbed than methane, as well as present in much smaller amount. Hence they are the last to be removed from the coal.

Absorption of Gases by Coal Dust

G. S. Rice (written discussion).—Professor Haldane has given greater attention to instantaneous outbursts of gas than I had appreciated. In fact I overlooked some of his discussion and that of others in the *Transactions* of the Institution of Mining Engineers [(1917) **52**, 338; (1921) **61**, 119; (1922) **62**, 298]. Although my attention was directed, in 1911, to the absorption of gases by coal dust and some preliminary tests were carried on by Dr. Thiessen, of the U. S. Bureau of Mines,[11] apparently the first

[11] International Conference of Mine Experiment Stations, Pittsburgh, Pa., Sept. 14–21, 1912. U. S. Bureau of Mines *Bull.* 82 (1914) 58. Compiled by G. S. Rice.

important work on absorption or adsorption by coal dust seems to have been that of Mr. Graham in so far as it related to mines.

The U. S. Bureau of Mines, about the same time (1917), in the development of gas masks for war purposes, carried on tests similar to those of Mr. Graham, with anthracite dust as the absorbent. Evidently, from the above discussion, many of the points which I touched upon briefly in my introductory notes had been considered. However, there appears to have been no general agreement, as indicated in the course of the discussion which appeared in the *Proceedings* of the South Wales Institute of Engineers in 1926 (**42**, 465). In my part of that discussion which appeared in 1927 (**43**, 32) I expressed essentially the same views as in the Notes.

Faults in Relation to Coal-mine Gases

Professor Haldane, in his present discussion, makes a reference to the disintegration of coal at faults. If by the word "fault" is meant true geologic fault with the shearing of the coal bed, I suggest that faults of this character do not generally produce a large amount of crushed coal. To obtain pulverizing of the coal requires lateral differential movement of the roof or floor of the coal bed. This is frequently found in coal beds in mountainous countries where there has been folding of strong sedimentary strata containing interbedded coal; the latter, being weaker structurally, takes the dynamic results of the lateral slippage. The effect in the coal bed is frequently erratic but most often there is rolling and local crushing of coal with some pulverization, yet adjacent to such points the effect may be to compact the coal.

When coal is crushed, as is well known, large amounts of gas contained in the pores of the coal are released. Generally under normal conditions most of the gas slowly escapes in the course of ages through the strata or laterally to the surface and these crushed places in coal beds give little more gas when mining approaches than other parts of the coal bed. On the other hand, under those rare combinations of conditions that result in the phenomena of instantaneous outbursts, the gases released by crushing of the solid coal are undoubtedly re-absorbed or adsorbed by the broken coal, especially by the pulverized coal recompacted by the pressure but not reunited structurally.

Previous discussions have suggested a limit of gas pressure comparable with a hydrostatic head from the surface, such as may occur with open or porous structure in a synclinal basin, but under the conditions of geologic thrust in deeply buried strata there would be practically no limit to the pressure under which the gases may be initially compressed. Recent tests in the experimental mine near Pittsburgh have shown that crushing of coal in place on a vertical surface by lateral thrust does not take place until the unit pressure is from 14,000 to 15,000 lb. per square inch.

In previous discussions of the outburst question, the effect of faults has been mentioned as a cause of coal crushing, but unless the term "fault" is meant to include shear movements in coal beds from lateral thrust, faults which shear across sedimentary strata and cause displacement usually do not crush the coal except marginally. My observation has been that usually normal faults drain gases to the surface or to contiguous porous sandstones.

Although Mr. J. Ivon Graham demonstrated the tightness of specimen coal slabs in 1917 in preventing the passage of air or gases, that does not seem applicable to practical conditions in a mine working in normal coal. Most coal beds have closely spaced joint places—"butts" and "faces," which, together with porous layers of fusain found in many coals, permit considerable degree of movement of gases. This is shown by the way the gas pressure drops as a mine working advances under ordinary natural conditions; 500 to 600 lb. gas pressure from a coal bed has been measured in deep boreholes from the surface, yet a mine blower rarely has a high pressure.

According to N. H. Darton, tests were made at St. Etienne, France, by holes bored from the mine workings into the solid coal. The maximum gas pressure in holes 23

ft. deep was 44 lb. In the gassy Agrappe mine (subject to outbursts), Belgium, Ghysen reports that the highest pressures obtained in drill holes was 15 lb. This was evidently under normal conditions of the coal bed.

Darton had test holes bored 10.5 ft. deep into the solid coal in certain southern Illinois mines, in which the gas pressure ranged from a few ounces to a maximum of 33 lb. The latter pressure was measured in a mine about 600 ft. deep to the level bed.

In the Coal Creek mine, Crow's Nest Pass coal field, British Columbia, in the course of the investigation of bumps and outbursts referred to in my paper, several holes were bored 24 ft. deep into the solid coal, where the coal was from 2000 to 2500 ft. deep and beyond was much deeper (5000 ft.) under a mountainous plateau. The maximum gas pressure found was but 23 lb. per sq. in., yet this mine was producing at the time over 4,500,000 cu. ft. of pure methane in 24 hours.

At that time (1917) this mine was not having outbursts but had experienced some terrific bumps which produced serious accidents. The cause of these was gradually cured by a change of mining method, but a few years later, when crumpled conditions of the coal were found, instantaneous outbursts occurred from time to time. Mr. Caufield, the general manager, has described these in his discussion. In this mine, geologic faults with displacements were not a notable feature, but the effects of lateral geologic thrusts were experienced.

In the Cassidy mine, Nanaimo coal field, British Columbia, where I investigated instantaneous outbursts of hydrocarbon gas in 1922 for the Minister of Mines of British Columbia, the coal bed dips about 20°, the roof is very strong and the floor is a clayey shale. The movement of the roof was probably up the pitch relatively to the floor, due to folding effects on a large scale. It had rolled up the coal and floor, crushed some of the coal in areas, and sealed the "dam" on all sides of the areas. Some of these places became outburst centers. Shear faults displacing the coal were not an observable factor.

It is therefore thought that faults are only incident in relation to outbursts, that at most they serve as channels from lower strata and in Lower Silesia permitted carbon dioxide to enter the coal. This gas presumably came from below, where volcanic intrusions were in contact with carbonate rocks. Whether outlets or inlets remain open from faults to outburst areas is as yet a question in the Lower Silesian mines. The outburst centers marked on the mine maps seem to have little or no relation to the faults.

Water as Factor in Outbursts

In regard to water as a factor in outbursts, Professor Briggs says:[12] "The entry of water for the first time into a virgin seam containing gas adsorbed under pressure—would bring about an evolution of gas. . . . " It is difficult for me to see how water could enter an outburst "nest" against an immensely superior unit pressure. I would look for pressure of the order of 75 or more atmospheres when the carbon dioxide and possibly some of the hydrocarbons might, through adsorption effects, be liquefied. In no other way does it seem possible to get the blasting effects by which hundreds of tons of coal, up to 6000 tons, have been blown from the face and enormous volumes of gases given off.

Absorption of Gas by Coal

J. I. Graham, Birmingham, England (written discussion).—The question of adsorption of gas by coal is one which has interested me since 1916, when at the

[12] H. Briggs: *Op. cit.*, 128.

suggestion of Dr. Haldane I commenced the experiments to which Mr. Rice has referred. Mr. Rice's interest and experience in this question of outbursts is well known although when I was conducting my early experiments I was unaware of the work carried out in 1911 by Dr. Thiessen at his suggestion, and of the fact that Dr. Katz was doing similar work. The work of Leprince Ringuet in 1914 was instructive but did not go far enough. I think that the investigations of Dr. Briggs on the adsorption of methane under high pressure and my own experiments carried out between 1916 and 1921, and published in the latter year, explained more clearly the relation of pressure to volume of gas adsorbed. Since then I have been in contact with Mr. George Roblings and have visited the scene of some of the unfortunate occurrences at Ponthenry. I am glad to note that Mr. Rice believes these outbursts are due to the presence of coal dust, produced as a result of earth movements, and recompressed in the presence of gas. I am convinced that this is the true explanation. One would think that the gas could be drained away through boreholes in such compressed material, but Mr. Greenwald's experiments (quoted by Mr. Rice) on the flow of gases through highly compressed coal dust show how fine dust can act as an effective sealing agent. Mr. Rice has referred to some of my experiments which showed how impermeable solid coal may be. My tests were carried out with one type of coal (hard and soft banded bituminous coal) and I am not sure that one will get similar results from coals which have been subjected to considerable earth movement. Further experiments with other coals are most desirable.

I agree that underground a coal seam at the face is, as a rule, far from gas-tight. My own attempts at measurement of gas pressures have shown that breaks occur 30 ft. into the solid and probably much farther, along which the gas can drain away, and no doubt the cleavage planes and layers of fusain help to lower the gas pressure in coal at a longwall face to a figure considerably below that in the virgin coal. Out of a great number of attempts to measure the gas pressure in boreholes driven in headings, or at the coal face in different seams, I never found a pressure greater than 40 lb. per sq. in. even at depths of over 2000 ft. below surface. Mr. Rice has pointed out that the pressure resulting from geological disturbances may be much greater than the hydrostatic head corresponding to the depth of the workings, but it does not seem necessary in the case of the majority of outbursts in this country, at any rate, for there to have been a pressure of much more than 10 atm. Our results for the adsorption of methane, combined with knowledge of the amount of coal blown out, indicate that such a pressure is sufficient to explain the large volume of gas evolved during the outburst.

In some of the Ponthenry outbursts, those occurring naturally and also those artificially produced by the method of blasting so ably applied by Mr. Roblings in South Wales, the blown-out coal showed a considerable rise in temperature above that normally found in the roadway, a temperature of 140° F. being noted by Mr. Roblings on one occasion. Several weeks after an outburst, I found a temperature of nearly 100° F. in the top layers of the coal dust which almost blocked a roadway 5½ ft. high. Such temperatures are due, of course, to the oxidation of the anthracite, which when powdered absorbs oxygen rapidly from the air. Unlike the lignitic coals, however, anthracite does not show a marked increase in rate fixation with rise of temperature until the latter gets well over 100° C.—consequently the danger of the dust firing spontaneously is not great, although a number of exceptional cases are known where anthracite has fired spontaneously.

Mr. Roblings and I hoped to get some records of the drop in temperature which undoubtedly must have taken place in the gases when first blown out in an outburst but unfortunately at the time of Mr. Roblings' tests we could not supply a suitable recorder.

Structure of the Coal

J. M. Royer, St. Martin de Valgalgues, France (written discussion*).—The paper by Mr. Wilson, with whom the writer talked during the course of the visit of the French Mission in Lower Silesia in October, 1930, following the disaster at Wenceslaus, sums up well the general opinion of the German and French workers concerning instantaneous outbursts of carbon dioxide. Nevertheless the writer wishes to describe in detail certain points, to set forth the conclusions drawn by the Mission from comparison of facts established in Lower Silesia and in other coal basins subject to instantaneous outbursts of methane and carbon dioxide.

All these basins are in formations affected by important tectonic movements; certain strata that at first sight appear to be undisturbed (Molières, France) have in reality been subjected to shift and the shales are marked by small undulations which demonstrate the lateral thrust movements of the beds in relation to their walls. These movements have caused a modification of the structure of the coal wherever there has been differential movement of the walls at the expense of the coal bed, as, for example: in the neighborhood of faults, where the displacement caused thereby has been oblique with reference to the plane of the bed; or in the mass of the bed, if it has been parallel to its plane (as in the Anton bed at Rubengrube, Lower Silesia, the Couche Sale at Molières, Gard Basin, France, the Chevalières bed of Borinage, Belgium, and the Douglas bed at Cassidy mine, B.C.); and locally if the crushing is limited to certain narrow zones (Ponthenry mine, South Wales).

Coal with this modified structure (*zermübte Kohle* of the Germans) does not absorb a greater volume of gas (CO_2 or CH_4) than normal coal, but it gives it up in an entirely different way. A bed of normal coal, breaking into cubes, loses its gas in a progressive and regular manner; a bed of coal of modified structure, on the contrary, gets rid of its gas with more difficulty but also more suddenly as the structure is more completely modified and more closely approaches mylonitic (amorphous) structure produced by dynamic metamorphism (of which the appearance is that of hardened slime), and which is the most dangerous (such as that in the Gard Basin and at Ponthenry mine).

Origin of the Gas

The origin of the CO_2 is almost certainly external: all the beds containing CO_2 are connected with recent gaseous growths, with or without the presence of eruptive rocks, which manifest themselves by mineral springs (Lower Silesia, Gard, Auvergne, Moravia) and which often have deeply attacked, by carbonation or kaolinization, the enclosing rocks. The permeability of these rocks has played a great part in the impregnation of the bed; the shales which are relatively tight have played the role of stopping the movement of gases through the sandstones of porous structures. Thus the beds between shales often are protected from CO_2 except when the displacement by faults brings it in contact with two porous horizons, which permits the passage of CO_2 along the plane of the coal bed, which then becomes very dangerous.

The outbursts of CO_2 are not confined to coal beds; there are outbursts in rocks which do not differ at all from those in coal (Brassac, Gard).

The impregnation of CO_2 takes place according to a law that appears to be intermediate between those of adsorption and those of dissolution: Greenwald's mathematical interpretation of his experiment probably may determine a formula for it. When a deposit normally impregnated with CH_4 is traversed by a flow of CO_2, this gas displaces progressively the CH_4 in proportion to the concentration of the two gases, without, however, being able to eliminate it completely. From that comes the well-known rule of beds containing almost pure CO_2, surrounded by a nimbus of mixed gas (CO_2,

* Translated from the French.

CH_4) which separates them from normal beds having pure CH_4 (Rochebelle, Trelys, Brassac, Moerisch Ostrau).

The origin of CH_4 in the beds subject to instantaneous outbursts is more debated; some attribute to it an external origin, basing their opinion on the distribution of explosive zones in certain Belgian deposits and on the presence, almost everywhere proved, of ethane and its higher homologues as found in the mine faces subject to instantaneous outbursts.

Also should be noted the instantaneous outbursts of CO_2 and mixed gas in the potash basin of the Werra and of those of nitrogen in the copper mines at Mansfeld, Germany, which show that these phenomena are not segregated in coal mines only.

Mechanism of the Outbursts

Little is known of the mechanism, because direct observations are rare and too dangerous, especially for CO_2. It may be said in every case that the phenomenon is not instantaneous and often continues during a fairly long time (Ponthenry, Cassidy, Gard). The risk of explosion of the CH_4 by shot-firing is therefore very slight and practically nonexistent.

All degrees are found in the same mine in nature and the appearance of the projections varies from the simple shoving outward of the face which comes out like a cake from a coke oven (Wanderer stoss at Pecs in Hungary) with slight gaseous emission, up to the free outburst liberating enormous volumes of gas, producing the mechanical effects of an explosion and accompanied by a large proportion of fine coal like soot (mill dust).

It is believed in France and Germany that the conditions necessary to the production of an outburst are a special aptness of the coal bed (as regards structure and amount of gaseous absorption), a strong initial pressure; it is also believed, in general that a mechanical action is necessary, which would be the means of unlatching the explosive phenomenon (mine blast, bump, work of tool—trigger effect).

The structure of the bed governs the speed of flow of the gas; this is the more regular and smooth as the coal is better divided into layers, but the more the coal is broken and of irregular condition, the less rapid is the gas flow; on the contrary, if the front of the coal face is overturned and if the cohesion of its structural elements disappears, the gas flow, which is progressive where pieces are concerned, is almost instantaneous when the particles are fine.

The initial pressure effect is to provoke the extrusion of the coal, which tends to move out for the length of the face of the working place like a plastic material forced through a drawing plate or die (example, lead pipe), and to modify the structure of the coal, which releases its gas the less progressively as it tends the more towards the mylonitic (amorphous) structure. This pressure may be of tectonic origin or may result from working of the ground. From this have arisen the following, which are the bases of the rules applied to exploitation of the beds subject to instantaneous outbursts:

To provoke a gas flow as strong and extended as possible from out the face of the working place in the solid mass of unworked coal; to reduce the ground pressure and to avoid creating an abnormal ground pressure and to regulate the mining method so that the speed of expulsion there is always below the danger limit.

The *spontaneous release* in the phenomenon appears scarcely possible; the progressive gas flow must augment the resistance of the front; there must therefore be an outside intervention to induce the release. This intervention may be provoked or involuntary.

The first case comes from the use of tools, which may be fatal to the operator, or it may arise from discharge of explosive (*tir d'ebranlement*) fired electrically from a distance under conditions of special safety for the shot-firer and other employees.

The second case is due to ground pressure or falls or slides which cause a crushing that destroys the protecting natural wall of coal and accelerates the gas flow beyond measure, or may be due to an earth shock or "bump" which acts on the face of the working place like a mine blast. The "floor shock" which characterizes the bump has been observed by the writer, at a safe distance from the board, in a face thrust happily without serious results, which occurred during the shift at Nord d'Alais colliery, and by two witnesses of the disaster of Wenceslaus (Lower Silesia); certain facts reported in the Gard basin tend to prove that the bumps sometimes are the consequence of certain blasting shocks and provoke "spittings" of the face at a fairly long distance from the point of explosion of the charge of powder.

Preventive Methods

The method of using drill holes to gage the gaseous content of the bed and to bring about a release of gas seem to have little value; maintained in Belgium and Germany by administrative order, it is considered inefficient by almost all the operators, for CO_2 as well as for CH_4. The gaseous flow does not seem to have any connection with the risk of outburst; there must be produced a clogging of the drill hole which is much more rapid so that the structure of the coal is more profoundly affected, and therefore becomes more dangerous. The decomposition of the coal in grinding a sample is slow and gives results of secondary interest; the specimen extracted from the face of the working does not possess the characteristics of coal *in situ.*

The only method of exploration that has some efficaciousness is that of the "blasting shocks," practiced during the absence of the workers, and in which shock is induced by heavy charges, in the face of the working place, in order to produce there an energetic gas flow. This method combats the instantaneous release by the release of the gas pressure; it is expensive and slow, but it is the only one that has permitted the working of the extremely explosive beds of Ruben (CO_2) and of Gard (CO_2 and CH_4) with comparatively few accidents.

Methods of Working

From the study of the mechanism of instantaneous outbursts, it is possible to deduce a certain number of rules to be observed in working:

Release of Gas.—Release of gas pressure is favored in the plane of the bed: in driving headings by blasting shock; in longwall faces by an appropriate swiftness of advancement, by mechanical cutting by a bar coal cutter (Lower Silesia) if the texture of the coal permits an efficacious gas flow and if the initial strength of the gas is weak; if the conditions are otherwise, by the blasting shock, used either in short holes as in France and Germany, or in long holes, as in Belgium.

Release of gas in the plane perpendicular to the bed is efficacious for a suitable thickness of strata (20 to 50 meters following the beds and the soils in working) above or below the dangerous coal bed, in the previous working of another coal bed called "shield bed," provoking at the same time the gas flow and the mechanical loosening of the dangerous bed.

Control of Ground Pressures.—Aside from this method which makes it possible to diminish the effect of ground pressures, an endeavor must be made to avoid everything that can develop abnormal pressures in the panel to be worked; such as residual pillars, blocks, projecting corners and steps of coal and all the irregularities of outline by which "crush bursts" may be produced and in a plane normal to the dangerous bed, avoid leaving pillar remnants in the shield bed, which may cause "punching bursts. In the longwall itself, one must avoid irregularities in the speed of coal removal, be careful of the support, fill up or provoke very regularly the fall of the roof in the gob,

align the faces and determine and maintain the proper intervals of working places, keeping count of the critical zones developed by the waves of pressure which follow the progressive advance of the longwall face. Moreover, it is necessary to regularly control the conditions of the ground to be exploited by headings of large dimensions such as the levels drawn by blasting and so avoid working a coal bed with an inadequate margin of safety without any attempt to investigate conditions.

The greatest danger is that of involuntary release of the phenomenon by a lateral ground movement or slide or by a bump. To prevent a rock slide in Belgium and Spain, they practice the lagging and buttressing of the front of the working face and also cutting to the dip (Belgium); in regard to the hazard of bumps the question is particularly difficult; Winstanley has recently shown (*Colliery Guardian*) the relative importance of the reactions which occur in the roof of a longwall in the course of the operations of cutting; they are little felt with a manageable roof but can become very dangerous with strong roof and floor. These reactions need not have the violence of those of Springhill or of Upper Silesia to provoke an overturning of the face and the extrusion of the complex of coal and gas which characterizes the instantaneous outburst (Molières and Gagnières, Gard). The inquiry into the disaster at Wenceslaus seems to have shown this well; the accident happened in advancing longwall in a 200 m. (600 ft.) length of longwall face. It progressed in a virgin bed, under a strong roof; the cutting of the coal was done by two bar coal-cutting machines working to the raise on 150 meters (450 ft.) length of face. The "aptness" to outburst of the bed was increasing, and also the rigidity of the roof; outbursts were being produced by shot-firing in the upper and lower level headings and in the higher part of the stope where the mechanical cutting had been forbidden by prudence. The accident occurred 30 meters from the lower end of the stope at the passage of the wave of pressure corresponding to the reaction of the pillar at the dip end of the face working at a point where they were mining out the coal that had been cut out by a machine about an hour before. The outburst occurred without a warning sign; the bump accompanying it was felt in the floor of the bed more than 800 meters from the place of the accident. The genesis of the phenomenon must be as follows: work on the longwall having practically ceased for nearly a month, the wave of pressure had moderated and the zone of abnormal pressure in which the structure of the coal was modified had come nearer to the face of the wall; the work of the cutting machine had provoked a first reaction of the roof which was followed, perhaps too rapidly, by the reaction to the stripping of the coal and that occurring exactly at the place of passage of the wave of pressure with relation to the pillar at the lower end of the face. This reaction provoked a first overturning of coal in the face, and caused a bump, which, acting upon an important surface of coal of modified structure, displaced *en bloc* a part of the face and uncovered the large channel through which the extrusion of the complex of high gaseous content was effected.

Action of Earth Pressure Important

This disaster seems to confirm the importance of the action of earth pressures in instantaneous outbursts. The gaseous characteristic naturally plays an essential and necessary role in the phenomenon and the end to be attained evidently is to provoke a release of gas as complete and efficacious as possible; but the role of the characteristic pressure, even if it is not perhaps always necessary, nevertheless often is sufficient to provoke the release of gas, and, because of this fact, it cannot be given too much attention. This role, foreseen by Herd and von Bubnoff, was made clear by the writer 10 days before the Wenceslaus disaster, in a lecture at the International Congress of Mines and Metallurgy, Liége, June, 1930.

Composition of Gases

L. Denoël, Liége, Belgium (written discussion).—We must confess that we progress very slowly in the knowledge of the origin of such phenomena and therefore valuable observations like those of Messrs. Rice and Wilson are welcome. The composition of the gas from Coal Creek is very different from all others, especially the Belgian gases, as the amount of ethane therein is exceptionally high. The samples recently tested from a number of mines contain only small proportions of gases other than C_2H_6. I suppose there might be also wide differences in the geological circumstances between our coal field and Crow's Nest Pass field. Outbursts are not identical in all districts in Europe.

Mr. Rice's diffusion tests and conjectures are of great interest. They include a strange fact. In Table 2, sample 8, the amount of C_2H_6 in the effluent gas shows a sudden increase from 0.3 to 2.9, yet the sum $CH_4 + C_2H_6 = 45.1$ is essentially the same as in the original gas. As Mr. Rice says, it requires caution to accept this result.

Mr. Rice's hypothesis of the core of outburst areas being filled with denser gases is akin to that of Mr. Ruelle, which supposes the existence of some chemical combinations of the form C_2H_6. It may account for some local circumstances, but presents great difficulties for the explanation of outbursts in coal fields where neither CO_2 nor C_2H_6 has been detected, either before or after an outburst.

I agree with Mr. Rice's conclusions regarding precautions, excepting the first, *viz.*, to block out the ground by levels and raises. Longwall workings may be better than the preliminary blocking out by levels and raises, as these workings on the whole are by far more dangerous than a continuous working face.

Absorptive Power of Coal

O. Ruff and E. Ascher, Breslau, Germany (written discussion*).—Mr. Rice makes several assertions that contradict the results of our experiments.

1. Mr. Rice draws from the values given in Table 6 of our first paper on outbursts of carbon dioxide[13] the conclusion that powdered coal is capable of absorbing considerably more carbon dioxide than compact lumps of the same kind of coal. We explained in our paper that the figures given for lump coal No. 9 in Table 6 must be considered minimum values, as when we used them we could not wait for the completion of the process because of the time required. Table 8, given in the paper mentioned, shows that in lump coal the condition of saturation could not be reached even after five days. Therefore we maintain our statement that the ability to absorb carbon dioxide is almost the same for powdered coal as for lump coal. The difference consists merely in the velocity of liberation of gas, which is high enough only in pulverized coal to explain the instantaneous course of the outbursts.

2. According to Mr. Rice, the ability of powdered coal to absorb carbon dioxide is essentially the same as that of activated charcoal. According to our experiments on this question (Tables 6 and 7, No. 12, as well as Fig. 18 in the paper mentioned above) the conclusion must be drawn that activated charcoal is capable of absorbing many times the amount of carbon dioxide found with coal considered as dangerous from the viewpoint of outbursts.

The Lower Silesian coals that were investigated had never shown the characteristic signs of activated charcoal; for example, a great ability to absorb phenol; neither could they be activated artificially by treatment with steam. After all, the question

* Translated from the German.

[13] O. Ruff, F. Luft und E. Ascher: Die chem. u. phys. Vorzänge bei Kohlensaüreausbruchen. *Ztsch. f. d. Berg., hütten- und Salinenwesen im Preuss. Staate* (1927) S46.

whether the carbon dioxide is bound in a dissolved or adsorbed condition is not important with outbursts of CO_2 because the ability of seam coal to absorb CO_2 is sufficient for the explanation of the amount of gas liberated during the outbursts.

Liquid CO_2 Not a Factor

Mr. Rice discusses the possibility of the existence of liquid carbon dioxide in deep earth strata. We do not doubt the possibility that occasionally there may be sufficient carbon dioxide pressure (about 45 atm. at 10° C.) in deep uncracked strata, but to compare the pressure of the liquid or gaseous inclusions in strata with the static rock pressure over them, as is done by Mr. Rice on page 81, is not correct. A pressure of about 45 atm. in strata and rock layers close to the region being worked is not possible because of its permeability to gas, and such pressure has not been observed anywhere. Therefore liquid carbon dioxide cannot take part in outbursts

Absorbability by Coal of CO_2

P. A. C. Wilson (written discussion).—Of the rocks interbedded with coal, shale is impervious, while sandstone is pervious to CO_2, and it may be observed that the beds of coal enclosed by sandstone are likely to have a larger amount of CO_2 and also to produce instantaneous outbursts.

It is important that the greatest amount of the gas is always in the neighborhood of faults, and the greatest danger of instantaneous outbursts is always there, too.

Carbon dioxide is found in mines sometimes in the form of blowers, or in solution in water entering the mine, but chiefly it is spread through the beds of coal in great areas. As a general rule, CO_2 is to be found in any coal beds of a number of Lower Silesian mines, especially Ruben and Wenceslaus, although instantaneous outbursts do not take place in all parts of the mines.

As to the way in which the gas is contained in the coal substance, it seems to be proved that carbon dioxide is absorbed in the coal. The tests mentioned by Mr. Rice indicate that coal is able to absorb rather considerable quantities of CO_2, the absorptiveness for CO_2 being about double that of methane for the same pressure. The volcanic CO_2 more recently given off from the depth on faults will certainly be under a high pressure. Also the beds of coal being under strong ground pressure, the coal substance in the veins is well able to absorb enormous quantities of CO_2 and even methane formed originally in the coal may be displaced by CO_2 on account of its stronger absorptive effect.

In Lower Silesia, CO_2 is supposed to be still rising from the depth, so it is believed that a stream of CO_2 regularly passes through the faults and even the beds of coal and pervious strata. Mr. Greenwald's tests, given by Mr. Rice, may well prove this supposition.

Since the absorptive effect for CO_2 rises with the pressure, it is comprehensible why outbursts of CO_2 became more frequent while mining was going to greater depth; that is to say, to zones of higher ground pressure. Up to 1925, but one outburst happened at a depth of less than 100 m. (328 ft.), 4 between depths of 100 and 200 m. (328 and 656 ft.), 130 between 200 and 300 m. (656 and 984 ft.), 179 between 300 and 400 m. (984 to 1312 ft.) and 123 between 400 to 500 m. (1312 to 1640 ft.) in depth.

Combination of Circumstances Producing Instantaneous Outbursts

In Lower Silesia, as elsewhere, there are certain doubts about the combination of circumstances that may produce an instantaneous outburst. A certain number of scientists believe, like the French, that outbursts of gas are caused either by rupture or by shocks imparted by the strata or by concussion of the blasts and other actions resulting from mining. Others hold the opinion—like the British, Mr. George S.

Rice of the U. S. Bureau of Mines and the author—that the places of outbursts were prepared ages ago by geologic movements, generally lateral, crushing the coal to dust in these places and that the gas was absorbed and concentrated in this dust. These points may have been eased by the greater surface of coal dust compared to lumps of coal. (In this case, coal dust found at outbursts is not the consequence but the cause of outbursts.)[14]

Another opinion is that the coal was not crushed to dust but only folded or shaken and that the gas was concentrated in the holes or vacant spaces made up in this way in the vein. In any case only these spots (prepared ages ago and not by mine working but exclusively by geological movements) which are called "clusters" or coal beds with CO_2 (*Kohlensäurenester*) in Lower Silesia, are the unique places where outbursts of gas are liable to take place. There are several reasons for this theory.

Bumps or shocks were sometimes supposed to have occurred in Lower Silesia but never absolutely observed. Further, according to laboratory tests, made up by Dr. Ruff,[15] coal in big grains or lumps will not give out the absorbed gas for several hours, while fine-grained—powdered—coal gives off the gas in several minutes. So it may prove to be the case with clusters of coal dust which are loose within the beds of coal. This dust is able to give out the CO_2 absorbed in it much more quickly than the uncrushed coal in place. A cluster is enclosed by an "embankment" or "dam" After this dam has been weakened or broken, the dust becomes able to give out the absorbed gas most rapidly—and that is the way in which instantaneous outbursts of gas take place. It is even thought that in the cluster carbon dioxide is in no larger volume or under a higher pressure than in the uncrushed coal—the sole difference being that the coal is of a different structure and therefore is able to give off gas rapidly. Moreover, there is another most important argument in regard to these clusters: As a general rule, outbursts of CO_2 are confined to certain parts of the mine—perhaps to certain veins. In the Ruben mine outbursts have taken place only in the middle part of the mine field, but in the south field and in the northern field no outbursts have happened, although the amount of CO_2 in the beds of coal is very high. In the same way, in the Wenceslaus mine outbursts were confined to the east field, the western field being free from them. As may be seen on the maps of the Ruben mine, the middle field is limited by two great faults, while smaller faults are found all over the field. Now there is a very peculiar condition of the beds in the middle field: There are wide zones in which the beds of coal are "rolled out," that is to say, beds become very thin and unworkable. For example, the Anton vein thins from 7 ft. to 1 ft. Probably these "rolls" were caused by geologic movements which may also have crushed or folded the coal.

Some scientists believe that the clusters are not prepared by geologic movements but by actions resulting from mining, especially by the "wave" of ground pressure caused by working which occurs within a certain distance of the face of workings and changes the structure of the coal. In Lower Silesia this opinion will not prove correct—if clusters are caused by secondary actions (such as mining) and not by geological movements, they ought to be found in any area of the mine and not to be confined to certain places. Moreover, they would become more frequent in beds in which working is going on. As a matter of fact, most outbursts have taken place in the newer districts, when driving headings or narrow places, and only 5 per cent. of the outbursts have happened in the subsequent wide workings, which may be termed panel longwall. Probably ground pressure effects caused by working would not be able to act over large areas under Lower Silesian conditions, because the many faults would change the character of the wave of pressure or stop it altogether.

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[14] *Coal Age* (1921) 209. [15] *Ztsch. f. angew. Chem.* **48,** 1038–1046.

Confinement of Clusters to Certain Parts of Field and Mine

Whether outbursts of CO_2 really are confined to certain veins is not known. Mr. Rother, chief of the Prussian Bureau of Mines, believes that in the bed worked first[16] outbursts will happen, when there are any clusters in the district—but that the veins worked next after a certain interval have "unbent" (relieving stresses) or else have given off gas under dangerous pressure after a lapse of time and may be worked without especial danger. This opinion is also held by the French and the question will be studied in Lower Silesia in future.

So it may be stated that outbursts of CO_2 may be expected from clusters, places prepared by geologic movements. The ways in which clusters burst out may be different: either the clusters are brought under an additional pressure by rupture, by shocks imparted by the strata or by concussion of the blasts or other actions (any kind of violent motion can cause an outburst) but always the supposition must be that there are present the clusters mentioned above.

In any case it may be observed that outburst clusters are usually confined to certain parts of the field and mine, while in other zones veins may give off CO_2, but are free from outburst clusters. This fact may give valuable suggestions in determining the best method of mining. Of course, the location of dangerous areas ought to be determined first by exploring. After exploring has been done, the character of danger of a district of the mine as a rule will be sufficiently settled.

Characteristics of Outbursts

There is one question which ought to be cleared up: Mr. Rice mentions that outbursts are similar in effect to great blasts of explosives, and he believes that the gas is held in the veins under very high pressure. I wonder whether this opinion will prove right for Lower Silesia: As far as I know, the British call "outbursts" sudden eruptions of mine gas from pin cracks torn up suddenly on the floor or sometimes in the roof of the longwall faces. By these outbursts large quantities of gas under high pressure are blown out, but surrounding coal and rock strata are not burst or broken but remain in their places. This kind of outburst is known especially in the mines of the district of Barnsley (Yorkshire). Similar phenomena have been observed in Westphalia (disaster of Radbod, 1908), but these outbursts have been only of methane.

The character of an outburst of CO_2 is quite different: At an outburst of carbon dioxide, the pressure of gas is rather small—about 3 atm. abs.—but the quantities of delivered gas are considerable and the speed of gas amounts to about 300 m. (990 ft.) per second. The effect of an outburst in the underground workings is far more similar to the effect of a strong hurricane than to the effect of an explosion. The timbering is scarcely destroyed. Coal and rocks thrown out seem to have floated or flown in the stream of gas. Heavy objects, like miners' trucks, are lifted and propped up with coal and coal dust. This may prove to be caused by rather low gas pressures.

In Lower Silesia, areas of extraordinarily high pressure of gas in the veins never have been observed. The ordinary pressure of gas in the wall 2 m. into the coal face is regularly less than 2 atm. abs. In outbursts, pressures of about 3 atm. abs. were indicated. This pressure seems to be not extraordinarily high, and never have zones of higher pressure been discovered by any exploratory holes. Cornet mentions pressures to 46 atm. abs. on 28 ft. from the open face observed in deep Belgian mines near Mons, but unfortunately he did not mention how the pressures were measured.[17]

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[16] Vollversammlung des Technisch-Wirtschaftlichen Sachverständigenausschusses für Kohlenbergbau beim Reichskohlenrat. Berlin, 1930.

[17] *Coal Age* (1930) 471.

Similar phenomena have never been observed in Lower Silesia. According to Dr. Ruff, outbursts are caused by the rapid deliverance of CO_2 from coal dust in the clusters, coal in this form being able to deliver gas more quickly than coal in lumps.

Any coal within a certain distance—probably to about 6 to 8 m. (20 to 26 ft.)—behind the face of the wall may be able to deliver the gas. The time of giving off of the gas will be dependent upon the structure of the coal. If a cluster or at least part of it lies in this zone of deliverance, coal dust will commence to deliver the absorbed gas, too, and will deliver it far more rapidly than the surrounding coal forming a natural embankment. The delivered gas will flow through the embankment and weaken the structure of the coal. The face may be forced outward and winning or digging by the hoe (adze-pick) may be done easily—until the embankment is weakened to a small dam which is broken even by a small pressure of gas. When blasting, the weakened embankment will be shaken much more than a normal part of a bed of coal; then the embankment may be broken on account of the shock and the outburst will take place.

I presume that the outburst is not accompanied by a shock or pressure like an explosion, but that it is more comparable to the breaking through of water. We know the effect of this in mines where great quantities of water are collected under small pressure. In several cases water has been tapped at the face of the wall only by a drill hole but the water streaming out has enlarged the small hole until the face has been weakened and broken and the water filling old workings has been able to break through violently. So I suggest that in most outbursts the gas will "break out" first in rather small places. This small hole will be enlarged by the stream of gas and coal dust of the cluster. So the power of the gas may increase and will be able to push and burst the embankment. The map and profiles of outbursts at the Ruben mine show that the channels of outbursts often are much smaller than the holes left from the outburst behind the wall. At Wenceslaus, I believe that the outburst commenced at one small place in the lower part of the cluster (near *B* in Fig. 8) and that the torrent of gas widened the channel until the quantity of gas delivered from the dust in the cluster became strong enough to push forward the block of coal rather easily.

Of course, all these events follow one another rapidly—in small fractions of seconds! At Wenceslaus, it was noted that apparently miners had not observed any change of conditions before they were killed, some of them being found in working posture and others sitting on their tool boxes eating—some men even were found with amused expressions on their faces. This observation may prove that outbursts are not accompanied by shocks or blows, and that the streaming out of enormous quantities of gas and dust follows instantaneously.

I hope that the questions of Upper Silesian outburst characteristics brought up by Mr. Caufield may be cleared up by these explanations. As concerns the term "cluster," it is meant to indicate a rather small place where gas (or minerals) are found underground—such as a nest, or group of nests or pockets saturated by CO_2.

Dr. Haldane asked for analyses of the return air under normal conditions. The analyses shown in Table 1 were taken in the Ruben mine in February, 1931.

TABLE 1.—*Analyses of Return Air in Ruben Mine, February,* 1931

Place	Air, Cu.M. per Min.	Men Working in This Air	CO_2, Per Cent	CH_4, Per Cent	O_2, Per Cent	Coal Mined and Hauled in 24 Hr., Metric Tons
Air shaft I.....	1060	103	0.68	0.02	20.14	240
Air shaft II....	1080	45	0.22	0.04	20.14	140
Air shaft III...	1040	116	0.74	0.04	20.00	240

The opinion held by Dr. Haldane, that the gas pressure in the patches of disintegrated coal which blow out is not any higher than in the adjoining coal, corresponds to the opinion of the German scientists, and so does his conception of the difficulties in measuring the true gas pressures. Tests to determine this question were attempted in Lower Silesia, but the results were not satisfying. Nevertheless, it is planned to renew these investigations.[18]

Doubtless the pressure of gas in the veins will be dependent upon the depth, the temperature of rocks, the ground pressure in the beds of coal, and also at least to a certain degree upon the pressure caused by working. In any case I question that in the clusters the pressure of gas will be extraordinarily high as compared with the gas pressure in its environs.

Mr. Rice says (footnote, p. 94) that pure CO_2 is odorless and suggests that another gas or substance may be present to give the odor mentioned. The "smell" of gas, as a matter of fact, is not an exact expression; the gas may be odorless, but it stimulates the pituitary membranes rather strongly and causes headache.

I was exceedingly interested in the report of Mr. Roblings on instantaneous outbursts of gas in South Wales. According to his contribution, outbursts on Gwendreath Valley are confined to one seam and only to the extreme west of the coal field. He mentions, too, that areas or belts, irregular in form, of friable coals have been met, and that the roof of the seam was formed by strong clift or shale. The conditions correspond to those of Lower Silesia, outbursts being confined to certain parts of the coal field. In Lower Silesia, however, shale in the roof and in the floor of the seam is supposed to be impervious to carbon dioxide which has entered the strata and the coal seams through faults from the depths below.

Now, if there should be sandstone in the floor or if the gas is CH_4, and was formed by geological processes in the seam itself, it may well be supposed that the roof stratum prevented the stratum gas from escaping even when the overlying seams were being worked. It would be useful to know the exact profile of the natural bed of coal and strata and the distances between the seams, and to know whether there are faults and other signs indicating geologic movements, although the phenomena described indicate that horizontal movements in the beds of coal were most likely.

The phenomenon of CH_4 outbursts and their results seems to be like outbursts of CO_2—especially it should be noted that CH_4 like CO_2 outbursts are in their effects to be compared to hurricanes.

According to the valuable tests made with telephones, the sound of firing and of small coal being thrown about was heard, but no noises of the main outburst. If violent outbursts should happen producing strong noises or shocks, probably the telephone or at least its diaphragm would have been destroyed, so the character of the CH_4 outbursts described by Mr. Roblings corresponds very well to that of the CO_2 outbursts according to the opinion held in Lower Silesia.

Mr. Caufield speaks of "bumps" causing outbursts. I do not know precisely his interpretation of the term "bump" but I understand it is a sudden sharp blow of the overlying rocks, caused by the roof breaking down over great areas at one time (like a shot), breaking the timbering, destroying any gates and working places and felt over wide areas, even above ground—sometimes like an earthquake. Bumps of this kind are known in Upper Silesia; in Lower Silesia they have been sometimes suspected but never exactly observed. Of course, breaking of the roof, especially when there is sandstone, over wide areas of goave is well known too. This may effect destruction of the timbering and also cause outbursts of gas. The breaking of the roof, however, usually goes on for a certain time—minutes to hours—while a bump occurs in a moment. According to Mr. Caufield, the roof in crushing down usually

[18] Untersuchungen über die Entstehung etc., 335. (Reference of footnote 1, p. 88.)

causes creaking, thudding, trembling and bumping. This is the ordinary way of breaking down of the roof. When this procedure is stopped, it may be that great areas of strong sandstone have remained hanging, and I suggest that the breaking down of these may cause violent movements of the ground and in this way cause outbursts. In Lower Silesia, gobbing or packing is done most carefully and working is done rather slowly to avoid extensive ground movement, which might cause bumps.

Warning Signs

Mr. Caufield thinks that boreholes are not of any assistance in giving warning of the likelihood of an outburst. I agree that boreholes are not useful for that or for releasing gas from the veins but they are useful nevertheless, in discovering the nature of a bed of coal in advance, such as the structure of the coal, the presence of faults and any other change of the natural conditions which will give rise to suspicion. So I would not say that we have had no success with boreholes.

The unsuccessful tests made by Mr. Roblings, mentioned by Professor Briggs, of injecting water through boreholes into the coal to prevent outbursts has also been tried in Lower Silesia, but the results were very unsatisfying and further trials were abandoned.

Warning noises, or at least something of that kind, were noticed in Lower Silesia formerly, before working in outburst coal was done by blasting. When the coal used to be worked by "hoes" (a form of flat-edged pick sometimes used in German coal mines, like an adze), it was claimed that in the neighborhood of outbursts the wall pressed forward into the open working place and that the winning could be done without any labor—it is even said that the miners were able to take the coal from the solid with shovels without any picking. In these cases a crushing and breaking in the coal was heard—"the coal was working by itself." The advance of the gateways or headings used to be very rapid. This natural extrusion of the coal was suspected as a warning. All working in outburst areas is done now by shock-blasting and the advance of the working places being rather slow, the phenomena described are not observed any more. Crushing and breaking of the coal sometimes may be caused by gas or by the ground pressure but are not certain signs of threatening outbursts.

Gas Clusters vs. Ground Movement as Causes

The discussion by Mr. Royer is of the highest interest. Mr. Royer has full knowledge of the French CO_2 mines and he has also studied the Lower Silesian outburst questions during his visit in this country in a very thorough way. His general remarks on the structure of the coal as well as on the origin of the gas may well be agreed with. Nevertheless, I understand that Mr. Royer does not agree with the idea of clusters of coal dust saturated with carbon dioxide formed by geological movements ages before, but that he gives great importance to change in structure of the coal caused by the ground movement from mining. Although a number of German scientists approve this opinion, I cannot accept this idea.

The question of outbursts may be considered from the geological standpoint and from the mechanical standpoint. Outbursts are confined to certain beds of coal and to certain parts of the mine, whether workings in these veins are in progress or only advance headings have been started, while in other parts of mines outbursts cannot be produced by any means, therefore I cannot agree in the mechanical view. In the northern part of the coal field of Ruben mine, for the first point of view all kinds of conditions which may surround outbursts are found: there are many faults, beds are flat as well as steep, coal is rather friable and the amount of CO_2 given off in mining the coal is rather high. Nevertheless, outbursts did not occur either in advance headings or in wide workings, although heavy blasting was done for many

years, even for some time by dynamite! In this part of the mine, breaking of the roof happened sometimes. In the middle part of the mine area, outbursts happened, although mining was done in the same way. So I believe in the presence of clusters of CO_2 being already in the coal veins which will burst out when mining is approaching sufficiently near in any manner, even when the coal of neighboring workings is delivering or releasing the gas from the respective faces in the ordinary way and without any difficulties.

The disaster of Wenceslaus was interpreted by some as due to mechanical actions, following a bump.

Gaertner[19] believes in a bump as the cause, because some men allege that they have observed a shock: "A short time after 4 p. m. the engine driver of the district traverse heading, 800 m. from the place of accident, claimed he felt a movement of the strata below him ('Ruck'). This was the bending move." This movement, it is theorized, thrust the block of coal from the face, and released small volumes of CO_2 which deafened the miners in the longwall first. Then the blow is supposed to have rolled farther on, crushed the coal behind the moved block of coal and CO_2 was released by the crushing, which smothered the men. I cannot agree with this idea. There is no reason to disbelieve the occurrence of a bump or a breaking of the roof, but the bump could not have been strong enough to have moved the enormous block of coal. Such a strong blow would have warned a wide area and CO_2 delivered afterwards rather slowly would not have found and killed the miners working or sitting in the ordinary way. By merely a bump or shock wave they would have become alert or would have started to run away. The positions in which the miners were found indicate that the outburst happened without any warning noises or shocks and that the volume of CO_2 delivered in a moment was great enough to kill them instantly.

Whether there was really a motion in the ground is difficult to say. The declarations of the saved miners ought to be taken up with greatest care.

I agree with Mr. Royer that the role of the characteristic pressure should be most carefully studied, as we know very little about it, but I am unable to follow the idea that ground movement from mining can cause outbursts in a bed of coal containing CO_2 if there are no natural clusters in the bed.

Reply to Discussion in General

G. S. Rice (written discussion).—It is gratifying to the authors of the original papers on instantaneous outbursts of gas in coal mines to have them so generously discussed by mining engineers and scientists of international reputation. The collective discussions provide valuable information from different standpoints and experiences on observed facts, laboratory investigation of different phases of the phenomenon, which is perhaps the most mystifying of any in the field of mining. Fortunately it occurs in comparatively few of the coal fields of the world and in a limited number of mines in those fields, but when it strikes, as in the case of the Wenceslaus mine disaster of July 9, 1930, the results are appalling.

The writer, in closing this discussion, will endeavor to answer inquiries made in the course of the discussion and to defend the views he presented. He will first explain, however, that he had no thought of not giving the fullest credit to those who have pioneered in this difficult problem, especially those who have had the most difficult task of all in trying to safely operate mines subject to these dangerous outbursts in Belgium, France, Lower Silesia, South Wales and, rarely, in other places in Great Britain, Western Canada and occasionally in other countries. The writer

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[19] *Glückauf* (1931), 149–186.

wishes to express regret that in the Introductory Notes he was unable to give proper acknowledgment to many who carried on research work and studies. Some of these will be referred to in this discussion.

When the U. S. Bureau of Mines was established in 1910, a piece of research work on the absorption of methane by coal dust was undertaken at the writer's request by Dr. Reinhardt Thiessen at the Pittsburgh laboratories of the Bureau. A report of progress was made to the International Conference of Mine Experiment Stations,[20] held in Pittsburgh in 1912. Circumstances compelled suspending this research. It was taken up again by Dr. S. H. Katz, whose findings were published in 1917.[21]

Neither of these researches reached the stage of determining the effect of pressure on absorption by coal. This, so far as known to the writer, who had the reference only recently brought to his attention, was first undertaken by M. Leprince Ringuet, of France. His report was published in 1914.[22] He carried on absorption tests of CH_4 and CO_2 for a number of coals, under pressures up to 80 atm. abs. Under the temperature (about 18°C.) the CO_2 was liquefied at about 40 atm. The volumes of gas in cubic meters per ton at 0°C. and 760 mm. pressures for the range of coals tested were respectively 8 to 12 cu. m. for CH_4 and 27 to 52 cu. m. for CO_2.

The important research by Graham on absorption in 1916–1917 was referred to; also that of Briggs, in 1921, on outburst coal with special reference to Ponthenry (South Wales) outbursts. A valuable discussion[23] followed the presentation of the Briggs paper, by Dr. J. S. Haldane, J. Ivon Graham and others. Mr. Graham described his research then in progress on the relative absorption of lump coal and dust and tests to determine whether gas-saturated lump coal would burst with sudden release of pressure. With a full release of valve in 15 sec. there was no tendency to burst the lumps. Dr. Haldane expressed the view that the absorption or adsorption of gases by coal was a solution phenomenon.

In a recent paper, Dr. Otto Ruff[24] expresses the opinion that "gases are not adsorbed on the surface of the coal substance but are dissolved within the coal." Dr. Ruff's absorption tests are most interesting and in general agree with those of Leprince Ringuet, Graham and Briggs. He goes somewhat further, making tests with highly compressed coal dust which he thinks prove that finely divided coal under high pressure may bring about the damming back of gases. In that respect he presents a view similar to that expressed by the writer in the discussion of the Ponthenry outbursts. It also may account, as Professor Briggs and the writer have commented, for the natural sealing, as if cased, of boreholes entering outburst areas or "nests." This is one of the most mysterious phases of the outburst phenomenon, that exploratory boreholes that have passed through or into so-called "nests" have found little or no gas and no high pressure manifested—with exceptions when the drill has been violently thrust back.

Dr. Ruff and Ernst Ascher, in the discussion on page 122, say that the writer has drawn the erroneous conclusion from their tests "that powdered coal is capable of absorbing considerably more carbon dioxide than compact lumps of the same kind of coal." They say that their footnote has been overlooked, explaining that figures given for lump coal No. 9 in their Table 6 must be considered minimum values, and

[20] Papers given at conference, compiled by G. S. Rice and published as U. S. Bur. Mines *Bull.* 82.

[21] Absorption of Methane and Other Gases by Coal. U. S. Bur. Mines *Tech. Paper* 147.

[22] Expérience sur l'absorption de gaz par la houille. *Compt. Rend.* (1914) 573, **158.**

[23] *Trans.* Inst. Min. Engrs. (1920–21) 132–36.

[24] Investigations of CO_2 Outbursts including Tests and Analyses. *Ztsch. f. das Berg- Hutten- und Salinenwesen* (1930).

that the condition of saturation could not be obtained even after five days. It is hoped that they will continue their testing to clearly prove or disprove their present theory that lump coal will absorb as much gas as dust from the same coal, which would mean that the surface adsorption phenomenon does not apply to coal particles; that it is entirely a solution phenomenon. Their view that coal dust has no more absorptive power for gases than lump coal seems at variance with the tests of Graham and Briggs. Evidently there is a field for further research.

They misunderstood the writer's statement about the absorptive power of powdered coal. He did not say that this ability "is essentially the same as activated charcoal." His statement was (p. 85): "That pulverized coal is akin to charcoal in its absorptive power for gases, particularly carbon dioxide."

The writer has no opinion to offer on the effect of activation of coal on its absorptive qualities but does believe that while all coal dusts will absorb gases, some have a much larger capacity than others, and this is indicated by tests. Whether coal from a "nest" in a coal bed is especially liable to outbursts if activated, he has no views. Samples of outburst dust from the Crow's Nest Pass coal field show no appreciable difference inchemical composition from coal elsewhere in the bed.

Dr. Ruff[25] makes a statement with which the writer concurs fully: "The capacity of the coal substance may be different in different coals and may vary within the same coal bed. However, in all coals it is sufficiently large to store the amount of CO_2 required for outburst."

Ruff and Ascher direct their third criticism to the suggestion put forward in the Introductory Notes of the possibility of there being liquefaction pressures of gases held in the ground. The writer's thoughts in this matter considered the possibility of these pressures obtaining: (1) by direct natural hydraulic pressure in deeply buried porous rocks such as sandstones; (2) by lateral tectonic thrust of deeply buried sedimentary strata, in which gas is imprisoned in a porous compressible stratum like coal. Under such conditions, the gas pressures may be enormous.

Gas Pressures

Apparently Ruff and Ascher are considering only the dead weight of the strata in referring to "static" pressure, but the unit pressure from this is enormous under loads of four or five thousand feet or more of rock under which coal in some places, at one time or another, may have been buried. Such resulting unit gravity pressures may be relatively small as compared with unit pressures from great tectonic thrusts in deep strata. Dr. David White, of the U. S. Geological Survey, has shown that the metamorphic changes in rank of coal are largely due to the great lateral arch thrusts. But even the lesser unit pressure produced by hydrostatic heads in porous sandstones may be sufficient for the liquefaction of CO_2 without proceeding to the surface tension phenomenon. Natural gas frequently is found in the United States at pressures over 2000 lb. per sq. in. (130 atm.) and there are several so-called "ice cream" wells producing CO_2 in Mexico and New Mexico, said to be under high pressure but whether sufficiently so at the temperature of the ground to liquefy or form snow is not known to the writer.

Whether or not gas is held under very high pressure in outburst centers or "nests" requires further research in the mine by new methods.

Normal coal-bed structure with its bedding and joint planes—faces and butts—permits its gas to escape freely to the outcrop of the coal or to faults which may release the gas direct to the surface or to porous beds like sandstones above or below the coal bed.

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[25] Reference on page 80.

When, however, the coal measures are overlain by clay or clayey shale, as appears to be the case in Lower Silesia, this may seal over the faults and prevent the escape of gas to the open air. In these mines there appears to be a network of faults more or less interconnected, possibly leading to the outburst nests or so-called cluster of nests. Yet the interconnection cannot be too great or a single outburst would drain a large territory. Judging from the mine maps such does not seem often to be the case, as successive nests are encountered by headings at short distances apart. Moreover, the drainage of gas from one bed does not appear to affect another. Nor does the free drainage of gas from the neighboring gases drain the next pocket perhaps only 20 or 30 ft. ahead. Hence the conditions in Lower Silesia, as concerns outbursts without reference to the difference in origin of the gases, seem akin to the conditions in the fields, Belgium or British Columbia, in which only hydrocarbon gases are found, and which, it is believed by the writer, are derived from the crushing of coal by the later geologic movement of a lateral character which at the same time has rolled up the coal more or less, forming "dams."

Formation of Outburst Nests

The formation of outburst places or nests in Lower Silesian mines and in the Gard basin of France is admittedly complicated, whereas the conditions are relatively simple in the Cassidy mine,[26] British Columbia, and as it is advisable to proceed from the simpler to the more complex, the Cassidy mine conditions will be described. There, a single coal bed is involved. It has a strong sandstone roof and clay floor. The bed is in a monoclinal structure dipping about 20° from an outcrop, normally giving ample release of fire damp from the coal. There has been a differential movement of the roof strata and the floor strata, from the general folding of the region, which has had no effect on the strong roof but has rolled up the clay-shale floor and coal bed, in a series of rolls more or less at right angles to the dip. The bed as found ranges from a few feet to 10 or 15 ft. thick. The coal has been more or less crushed and the particles are slickensided.

The outburst "nests" generally are formed with hard dense coal acting as the restraining dam. Outbursts did not begin to occur until a vertical depth of about 1000 ft. from the surface had been attained. Undoubtedly nests higher up had been drained gradually through the geologic periods, as the anticlinal arch had been eroded and the outcrop lowered more and more.

As the mining advanced down the slope and the workings extended on either side, the outbursts occurred with increasing frequency—about three per month; most of them small and giving warning in sufficient time, as the dam gave way, for the men to escape. But in the course of a couple of years a number of miners had been killed and the morale of the employees affected.

One outburst of great size occurred in 1922, in which about 4000 tons of coal was more or less broken and much dust and fire damp filled the return air for many hours. In this case, which was investigated by the writer, the irregularity of the outline of the outburst area indicated a series of connecting "centers" which in Lower Silesia would be termed a "cluster of nests."

Origin of Gases of Outbursts

The writer reaffirms his belief that the hydrocarbon gases are derived from the pulverizing of the coal which has reached at least subbituminous rank, or a part of the vertical section, by the same lateral tectonic movement that has made the more or

[26] See R. R. Wilson and R. Henderson: Outbursts of Gas and Coal at Cassidy Colliery, Vancouver Island, British Columbia. *Trans.* A. I. M. E. (1927) **75**, 583. Mr. Rice is not quoting from this article, but speaks from his own observations.—Ed.

less tight surrounding dam. Doubtless the gases from most of the "nests" so formed has leaked away through past ages and only a few nests retain the gases. In the escape of the gases those of lesser density—methane and nitrogen—would probably have escaped first, so there would be a tendency toward the accumulation of the heavier hydrocarbon gases and CO_2 found on crushing coal, therefore gases of varying composition might be found in a nest, but this is conjectural as yet, except as indicated by the test work on diffusion of gases given on pages 81–84 in the Introductory Notes.

The origin of carbon dioxide as found in such great quantities in the Gard and in Lower Silesia is undoubtedly from igneous intrusions below the coal measures. These intrusions coming in contact with carbonates would naturally cause the distillation of CO_2. The forcing in of sheets is very likely to cause faulting and so it was in these cases, therefore the gases would rise through the fault planes to enter the porous strata such as coal in its bedding planes and where there was crush and fine grinding by lateral movement; with confinement of clay strata overhead, the CO_2 would be likely to be under pressure and would be likely in places to be mingled with fire damp. Under these conditions large amounts would be absorbed by the pulverized coal dust. If there were any leakage it would be of the fire damp first. If this is the way outburst centers are made, one might expect a great range of pressures under the different conditions of formation.

Pressure and Volume of Gas

Dr. Ruff, in his several published reports, has indicated that CO_2 is held under a pressure of only 2 or 3 atm. abs., 15 to 30 lb. per sq. in., gage pressure. Mr. Wilson describes what he terms the effects of a hurricane and mentions, in his discussion, a velocity of gas of 300 m. (990 ft.) per second. Such a velocity would be equivalent to that of the gases of a strong coal-dust explosion. It would be a speed of over 700 miles per hour, which is far in excess of recorded hurricane velocities which have caused destruction. To obtain such a velocity in a mine passage would require a velocity-head far greater than 1 or 2 atmospheres.

Mr. Wilson questions the comparison that the writer makes of the outburst effect of an explosive. Admittedly, there are all kinds of outbursts as regards pressure and volume of gas. The writer had, for a typical outburst, considered that the effect was that of a quantity of slow-acting explosive, such as black blasting powder in a deep hole well stemmed, but not a detonating explosive.

For the past few years the Bureau of Mines has carried on tests on the strength of concrete mine stoppings in a chamber in its experimental mine, about 700 ft. from the mouth of the mine. Charges of black blasting powder were exploded in this chamber, of about 1000 cu. ft. in size. The gases pressed against the stopping. The charges were successively increased, in some cases up to 25 lb. of black blasting powder, to produce a pressure of about 100 lb. or more on the test stopping. Until the stopping was ruptured there was no sound heard at the mouth of the mine, either through the ground or air. When the stopping was ruptured a loud boom was heard.

There have been outbursts in the Cassidy mine in the face of a heading by which a plug of coal of practically the full section of the heading was thrust up a steep grade, doubling up the mine track. This, it appears to the writer, requires a relatively high pressure though small quantity of gas. A low pressure, regardless of the quantity, would not do it.

In the disaster at Wenceslaus, Mr. Wilson conjectures that the sequence of phases of the outburst was: (1) the blowing of a hole through the coal at the lower (dip) end of the area affected, (2) a torrent of gas following, which moved a block of coal 53 ft. long and 26 ft. wide a distance of 6 to 10 ft. without crushing it.

If the pressure of the gas was but 2 or 3 atm. abs, equal to 15 to 30 lb. gage pressure, this seems impossible, regardless of the volume of gas behind it. Equally incredible

is the suggestion coming from other German mining men that a ground wave from a distance or a bump caused the moving of the block. A great bump might easily smash such a block, many examples could be cited, but to move it would require the wedging action of the upheaval of the floor or bottom into the coal in such a manner as to give a lateral thrust toward the goave. No evidence of this is indicated in the map and sections or in the description.

Mr. Wilson's observations of the effects of the outburst on the miners in the vicinity is of great interest. It seems probable that miners would start to run when a quantity of CO_2 gas reached them while their lungs were still filled with air; CO_2 in its effects is not like an insidious poisonous gas, such as CO, or even an inert gas. It causes rapid breathing in the first stage. The usual action of men, following explosions, the writer has observed, is to start to run even when the air is unbreathable. It is when an explosion has caused a severe shock wave, followed perhaps a minute later by poisonous gases, that men are found at their tasks or rest, as the case may be. In regard to Wenceslaus, it seems possible that the men found as Mr. Wilson describes were shocked by a high-pressure wave as the block, acting like a stopping or dam, suddenly sheared from the roof, floor and adjacent coal.

What gas pressure would cause the block to be sheared from top, bottom and sides is a problem, but it does not seem to the writer, from observation of explosion effects in mines and in testing, that it could be less than 10 atm. (150 lb. per sq. in.) and might be far more than this.

Effect of Bumps in Causing Outbursts

Mr. Wilson asks about the effect of "bumps." If the alleged bump did occur prior to the Wenceslaus disaster, it should have been felt on the surface. While bumps may vary in magnitude from that which gives the faint "boom" of a distant fall, what are usually termed "bumps" are sufficiently violent to smash timber or coal pillars. The writer, in his report to the Minister of Public Works and Mines of Nova Scotia[27] has attempted to classify bumps as "pressure-bumps" and "shock-bumps." The former occur when a pillar is too weak and, crushed between floor and roof, flies apart. The shock-bump is a shock wave caused above mine workings generally by subsidence, making wide openings under massive ledges of rock. When they fall, if it is a matter of a mass of rock weighing hundreds or thousands of tons striking on rocks below, a wave is set up which may also smash pillars and timbers, but quite usually if the roof is strong that is unbroken.

The Coal Creek mines have experienced both gas outbursts and bumps, and sometimes together. Mr. Caufield in a recent communication, says that on June, 2, 1931, they experienced the most severe bump since 1917, but no gas was given off.

Outbursts of Gas in Other Mines than Coal Mines

Ministerialrat Rother, Chief of the Prussian Mining Department, has kindly informed the writer that outbursts in Germany have occurred in mines other than coal mines. He says: "With the nitrogen outbreaks in Mansfeld ore mines as well as with carbon dioxide outbreaks in potash mines of the Werra district, one deals with standard (*regelrechte*) gas outbreaks. That is, with a phenomenon in which considerable amounts of gas are thrown into the mine suddenly and simultaneously with large amounts of fine dust. The gas outbreaks of the Mansfeld ore mines and in the potash mines of the Werra district are similar to those in coal mines, due to fire damp and carbon dioxide outbreaks, with regard to their outward appearances, and no doubt

[27] Reference of footnote 2*a*, page 77.

they are completely similar to these in character. The amount of rock masses thrown into the mine amounted to 600 cu. m. in the Mansfeld mines and 500 cu. m. in the Werra district."

It seems probable that such outbursts of gas in mines other than coal mines must arise from gas stored in open cavities. When details are available, study of them may throw light on some characteristics of instantaneous gas outbursts in coal mines. It is important to know whether the bursting of the rock or mineral was not due to shock-bumps or rock-bursts under pressure, as in the Rand mines and Lake Superior mines, with the issuance of gas as a secondary effect not necessarily under high pressure.

Indications of Pending Outbursts in Coal Mines

Professor Briggs has asked for further information about noises or other indications preliminary to outbursts. Those who have had practical experience agree that in many cases there has been little warning, but judging by the escape of men at the face in most instances of outbursts, there are symptoms a moment or a few moments ahead of the final rupture. There is the instance, however, of the great outburst at Wenceslaus, where men, according to Wilson, were stricken at their tasks or while eating lunch, and all in the face perished. On the other hand, there was a very small outburst in the Cassidy mines where eye-witnesses escaped. Only a ton or two of coal and dust was blown out, with little gas, yet two men were smothered. It occurred in a heading in new territory where there was no wide work in the vicinity. The heading was in thin coal, 2 or 3 ft. thick, and the shale floor had been lifted for a thickness of 3 or 4 ft. One man was working between a mine car and the rock bench and the other victim on top of the bench was mining the coal with a pick. A surveyor for setting "sight lines" had his transit placed about 60 ft. from the face and his flagman stood at the outbye end of the car referred to. They heard a blowing sound and the coal and dust were thrown out, with the man on the bench. Dust filled the air. The flagman and surveyor were able to get away and presently when the air had cleared somewhat they came back and dug out the miners. The latter had smothered, chiefly, it was thought, owing to the fine dust, which choked them.

In other instances in various countries usually the miners have been able to escape. In characteristic cases, a few minutes before the outburst there may be a little flicking off of fine particles from the roof and ribs; then after a brief pause, when all is very quiet, there is a slight movement with small noises (which perhaps is the indication of shifting of the rock stresses or beginning of failure); then may come the noise as of a riveting machine, which is notice for men to scramble for the nearest intake. This may be followed by more or less crashing and wind effects, enough to blow out timber and cause falls of severity. It is probable that no two cases are just alike.

Draining of Gas

The use of shock-blasting, firing from the surface or at least from a safe place, in the writer's opinion is the best means of combatting the hazard of outbursts so far as is known, and he so recommended to the Minister of Mines of British Columbia in 1922 in an unpublished report. The Minister of Mines feared that an outburst of fire damp might occur between the time of placing the charge in the borehole and electric firing from a distance, in which case the explosive might be fired after it had been thrown out of the hole by the outburst. The writer contended that even if this occurred, the outburst of gas would make a mixture with air beyond the explosive limit and, at the worst, if ignition of the gas did occur, the men having been withdrawn to the surface or behind a strong air lock, there would be no hazard to life.

In shock-blasting in the several countries no case has been reported of the explosive having been expelled from the hole by gas outburst.

The writer proposed, page 86, going further in the development of shock-blasting. Instead of holes only 6 or 7 ft. deep, as now customarily used in the several districts in different countries, he proposed drilling the advance holes at least 20 ft. deep and blasting with large charges after stemming with rock dust. Probably fewer holes would be needed. The objective is a little different from that of ordinary shock-blasting with relatively shallow holes.

The testing of the strength of stoppings in the experimental mine near Pittsburgh, Pa., has developed (1) the great unit strength of coal in place and (2) the great strength of an arch, though it be in the outward form of a slab. An outburst "nest" has, as a natural "stopping," a "flat arch" dam of coal which restrains it from bursting out. Observations have indicated that for average conditions in a heading this coal "dam" is perhaps less than 6 or 8 ft. thick to the gas-saturated dust or soft coal characteristic of an outburst nest; before the restraining dam shears or ruptures. Hence the proposal of the deeper holes so loaded as to have a shattering effect at their inner ends which may release high-pressure gas that will perhaps escape through the holes without destroying the "dam" and causing other destruction. This form of release of gas pressure is analogous to the common practice of shooting a gas or oil well in tight sands to obtain a flow of gas.

General Review of United States Bureau of Mines Stream-pollution Investigation*

By R. R. Sayers,† W. P. Yant‡ and R. D. Leitch, § Pittsburgh, Pa.

(Pittsburgh Meeting, September, 1930)

In 1924, the United States Public Health Service was requested to undertake a special study of stream pollution. The Public Health Service asked the United States Bureau of Mines to take up the study because the training and experience of the personnel of the former was considered not to be as well suited to this type of investigation as that of the Bureau of Mines. Under H. Foster Bain, then Director of the Bureau, field investigations were begun in the spring of 1925. Along with mine-drainage wastes, the problem of gas-house and coke-oven waste was investigated in the laboratory and a method of treatment by sodium hydrate and benzol was found to be promising. The idea was obtained by A. C. Fieldner, chief engineer of the Bureau Experiment Stations, on a trip to Germany in 1924. He saw a practical application of the process, but obtained no information as to definite chemical or mechanical processes. Because of the then pending American patent applications by the German firm, we made no attempt to develop our findings further. Shortly afterward, a similar method of treatment was developed in the United States,[1] and since that time we have devoted our attention entirely to mine drainage. This paper is a presentation of some general facts and information gathered during the past five years on coal-mine drainage.

Scope of Work

There was such a diversity of opinion at the time this work was begun, on the whole subject and apparently so little accurately known, that it was decided to undertake a basic study from which first, certain definite information might be evident, and second, what might be considered

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* Published by permission of the Director, U. S. Bureau of Mines.

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[1] H. E. Jones: Phenol Recovery Plant Avoids Waste Pollution of Streams. *Chem. & Met. Eng.* (1928) **35**, 215–218.

R. M. Crawford: Elimination and Recovery of Phenols from Crude Ammonia Liquors. *Ind. & Eng. Chem.* (1926) **18**, 313–315, and (1927) **19**, 168–169.

side issues of the original investigation would be suggested for subsequent study.

A stream in a district thought to be representative of those in low to average sulfur bituminous coal fields and another in a high-sulfur district were selected for study. A fair number of factors entered into the tentative selection of such streams, the chief ones, in addition to the presence of mine drainage, being their accessibility and freedom from other industrial or domestic pollution. Fig. 1 shows a typical stream and Fig. 2 a map of a stream that was sampled.

Fig. 1.—A typical stream in a mining district.

We have attempted to determine the variations of mine-waste waters as to acidity in the two districts, in different mines in the same district, and in different parts of the same mine. We have also attempted to determine the seasonal variations in mine drainage and the resultant effect on streams. Attention was also given to the possibility of sealing abandoned or worked-out mines, or parts of mines and influence of outside "gob" piles. These data were reported[2] by the Bureau of Mines in 1926 and 1928. The apparent recovery of naturally caved mines so far as acidity of drainage is concerned has been treated in *Report of Investigations* 2895, made available in 1928. At present, the relation between the amount and occurrence of pyrite in coal, top and bottom rock and gob material is being studied, both in the field and laboratory.

Non-acid Mines

It is commonly supposed that coal mines give off acid waste waters. Generally speaking, this appears to be true, but at least one large and important coal area has little acid drainage.[3] Some time ago it was learned that a mine in the Thick Freeport area had no acid water. The main field of the Thick Freeport bed is in Allegheny County, Pennsylvania,

[2] R. D. Leitch: Stream Pollution by Acid Mine Drainage; and Observations on Acid Mine Drainage in Western Pennsylvania. U. S. Bur. Mines *Rept. of Investigations* 2725 (1926) and 2889 (1928).

[3] Unpublished report, U. S. Bur. Mines.

an area roughly rectangular in shape and about 11 by 17 miles in extent. Detached areas are said to be in the vicinity of Argentine and Hilliards in

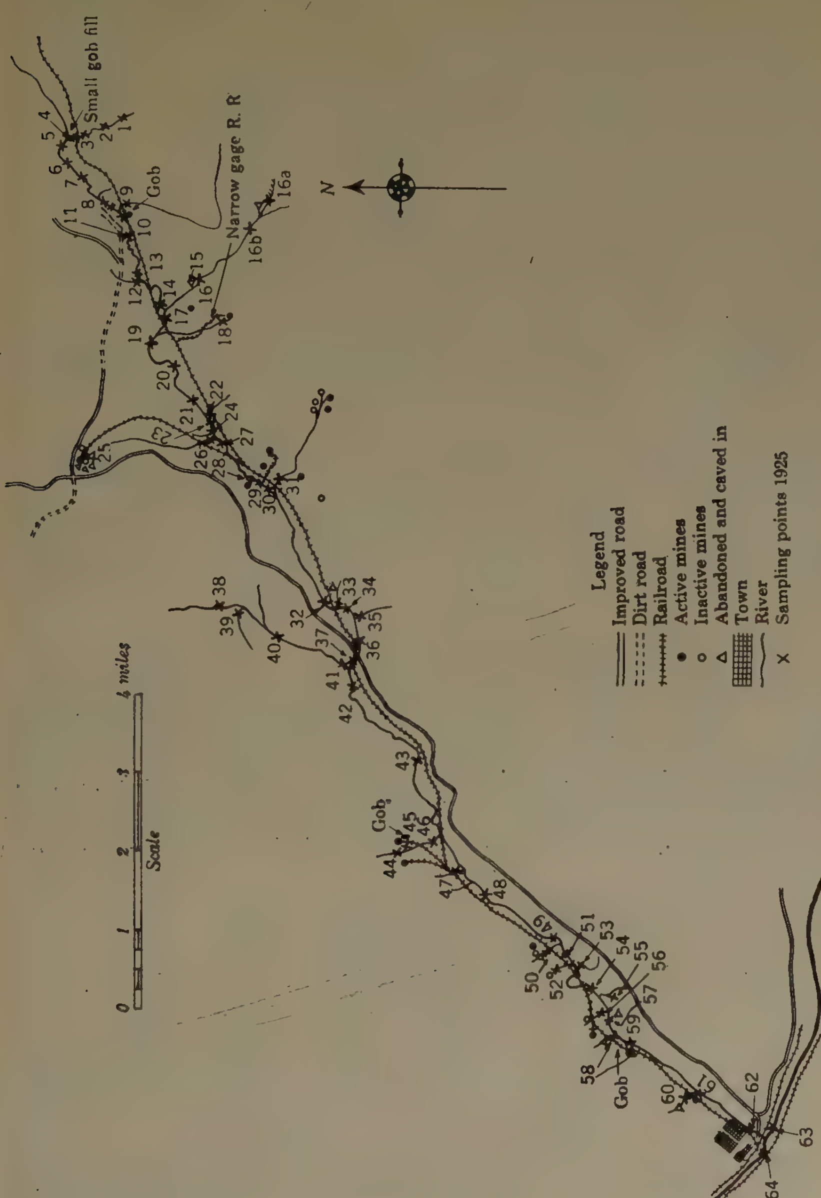


FIG. 2.—MAP OF STREAM THAT WAS SAMPLED.

Butler County, Coral and Graceton in Indiana County, and Derry and Latrobe in Westmoreland County. Most of the mines in the main area and several in detached areas were visited, and samples of coal, top and

bottom rock and gob, and water were collected. Of 17 mines so visited and sampled, 6 had no acid water, 6 had both acid and alkaline water, and of those having acid water, only one was strongly so. Many samples were very highly alkaline, drippers especially had alkalinity values as high as 1200 parts per million. This is many times as high as normal surface waters and, as an example, one good stream in southwestern Pennsylvania (Indian Creek) is said to have an alkalinity of 17 to 19 p.p.m. The Thick Freeport bed is a low-sulfur coal and has about 0.6 per cent. of the total sulfur in combination as organic sulfur, which does not enter into an acid-forming reaction. Waters that enter mines in the Thick Freeport bed have an unusually high alkalinity and therefore are able to neutralize considerable amounts of acid or inhibit its formation. There must be a bed of highly alkaline material with which these entering waters have previously been in contact, but core drillings rather widely scattered over the field do not show any unusual conditions other than that the strata are irregular.

In addition to these mines in the Thick Freeport bed, others in Pennsylvania[4] and other states[5] have been found not to have any acid water. Sometimes this is due to limited development and light cover, but there are unquestionably many mines which, for other natural reasons, do not make acid water.

Sealing Mines

During other work we have observed many abandoned and caved mines or otherwise sealed, from which effluent water was either not acid at all or only slightly so. In a few cases,[6] water from such mines has been used as a source of supply for domestic and drinking purposes. Most of these have been in districts and coal beds where water from other adjacent and active or open mines has been acid. There is no proof, of course, that these sealed mines were ever acid, but evidence is strong in support of that condition. From the fact that the accepted theory of acid formation in coal mines requires the presence of water, oxygen and iron sulfide together, it would naturally follow that if any one of these three constituents is absent no acid will be formed. In the past, we have endeavored to find some nearly worked-out mines, so that observations might be made on them both before and after sealing. Such mines must have sufficient cover at all points to prevent breaks to the surface, through which "breathing" might effect a considerable interchange of

[4] R. D. Leitch and W. P. Yant: A Comparison of the Acidity of Waters from Some Active and Abandoned Coal Mines. U. S. Bur. Mines *Rept. of Investigations* 2895 (1928).

[5] R. D. Leitch, W. P. Yant and R. R. Sayers: Effect of Sealing on Acidity of Mine Drainage. U. S. Bur. Mines *Rept. of Investigations* 2994 (1930).

[6] Observation of the writers and information given by other observers.

air, even though the entrances were sealed. It has been impossible, so far, to find suitable mines. In 1928 our attention was called by W. S. Harris, superintendent of the Panhandle Coal Co. at Bicknell, Ind., to a mine in the southern part of that state. He said that water from certain sections of their mine before sealing had been highly acid, as shown by excessive corrosion and consequent replacement of acid-resisting pipe lines and pumps. Shortly after sealing these sections the water was said to have become clear and no repairs were necessary during four years to date. Mr. Harris said that no explanation of these observed facts had been apparent to them until reference was made in the magazine *Coal Mine Management* to a report of the Bureau of Mines in which sealing was advocated to decrease acidity. He was immediately interested in the probable value of the information to the coal industry and asked the Bureau to send a representative to sample water in his mine and observe reported conditions. A visit there verified his statements and a few months later seven other mines in southern Indiana were visited and sampled in a similar way. This work was reported in detail in Bureau of Mines *Report of Investigation* 2994, already cited. Briefly, the results were: (1) that no sample of water from behind seals contained free sulfuric acid and only one had sulfates of iron in solution; (2) every sample in open sections of five mines, of long standing, was acid; highly acid, as a rule. Finally, three of the eight mines visited apparently had no acid water in either open or sealed sections. As a rule, gas samples from behind seals in these mines show the presence of oxygen to be less than 1 per cent. The mines were selected at random and solely upon information from the local district mine inspector that samples of water from both open and sealed sections could be taken in these mines. Three different coal beds were being worked in the group selected. They were all shaft mines from 265 to 315 ft. in depth and had the room-and-panel system of mining.

Laboratory Tests on Iron Sulfide

Laboratory tests have been made[7] using pyrite and marcasite in specially designed and constructed apparatus, which show the marked difference in acid formation as between oxygen and inert gases. In these tests moist air, as well as moist natural gas, was repeatedly passed over the same iron pyrite. The moisture was condensed and acidity determinations were made. Condensate from the moist air tests was invariably highly acid and that from moist gas was extremely low. No particular effort was made to prevent contact with air while washing the samples and replacement in the apparatus prior to making gas tests. The amount of acid formed in the latter was therefore assumed to be due to previous

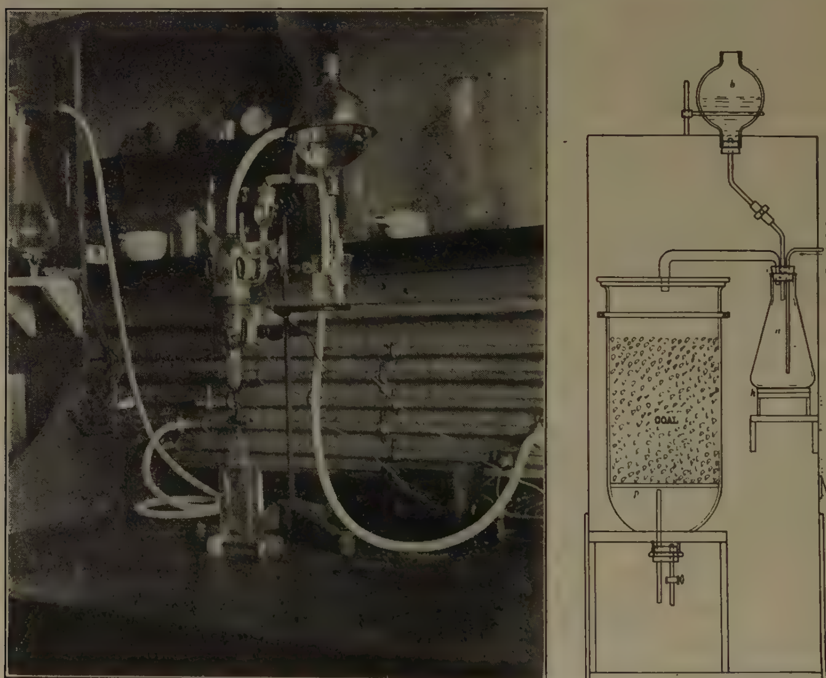
[7] Unpublished reports, U. S. Bur. Mines.

contact with air and possibly (though not probably) to very small amounts of oxygen in the gas, as no attempt was made to remove it, if present.

All of our work points to the fact that, where possible, exclusion of air from mines or parts of mines will result in preventing formation of acid. Fig. 3 shows the laboratory apparatus.

Effect of Mine Drainage on Fish

In many ways a consideration of the effect of acid mine drainage on fish has no place in this discussion; at the same time, and realizing the presence of many sportsmen in the mining industry, it has been thought



a *b*

Fig. 3.—Apparatus for oxidation of iron pyrite.

of enough interest to include references thereto. Also, various organizations of sportsmen have naturally the greatest interest in mitigating stream pollution in general and at present mine drainage in particular.

There is a general lack of knowledge as to just how fish are affected by acid drainage or how much can be tolerated by them. Communications from Fish and Game Departments of several states, the United States and Canada, are in general agreement that more than four parts per million of mineral acids will kill trout and that this fish is less resistant

to pollution than any other. Carp and bullheads are most resistant, with the so-called game and pan fish occupying various places between these extremes.[8] A lack of oxygen can and does kill fish, as well as direct toxic effects of the polluting substances.

Laboratory experiments have shown that one volume of water having an acidity of 1000 parts per million caused by the addition of ferrous sulfate can reduce the dissolved oxygen saturation value of from four to five volumes of pure water to zero. Field observations[9] have shown that entry of mine drainage into a stream having an oxygen saturation value of 40 per cent. can reduce the whole to a value lower than 30 per cent. Although rapidly flowing streams can quickly build up a reduced oxygen value if uncontaminated further, the temporarily lowered value may kill fish. Our work, however, has shown[10] that fish most likely are killed by the irritating effect of acid on the gill follicles, rather than by any lack of oxygen.

It has also been shown by Wells, Forbes and Richardson[11] that deposits on the bottoms of streams as a result of previous pollution of various kinds inhibits fish propagation by disturbance in egg laying, even though the water alone may be entirely suitable. It is problematical, therefore, whether the purification of streams once polluted will make them suitable for fish life for an indefinite period.

Mines Ordered Sealed

The so-called Indian Creek case in Fayette County, Pennsylvania, has attracted wide attention and is perhaps familiar to most coal-mining men in this district. Reference to it is made, however, to illustrate the seriousness of the mine-drainage problem in certain instances at least.

In 1927, a court order was issued enjoining 29 operating companies in Indian Creek Valley from permitting mine drainage to enter the waters of Indian Creek above the Pennsylvania Railroad Company's dam. This came as a final result of about seven years of legal battles through the courts of the state, in which there was some of the best legal talent on both sides and volumes of expert testimony were introduced, both for and against the mine-drainage question. For some months prior to the final settlement, three of these companies operated chemical treatment plants, where drainage from their mines was treated with sufficient lime to neutralize the acidity of the water. Subsequently, the iron was removed by allowing the treated water to stand in shallow ponds for several

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[8] M. M. Wells: Reactions and Resistance of Fishes to Carbon Dioxide and Carbon Monoxide. *Bull.* Ill. State Lab. of Natural History (1918) **11**, 557.

S. A. Forbes and R. E. Richardson: Some Recent Changes in Illinois River Biology. *Ibid.* (1919) **13**, 139.

[9] Unpublished report, U. S. Bur. Mines.

[10] B. G. H. Thomas: unpublished report, U. S. Bur. Mines.

[11] Reference of footnote 8.

hours, to settle out as iron hydrate. While satisfactory so far as producing a clear and alkaline effluent was concerned, the process was said to be expensive and caused no little trouble. By so treating the drainage, these companies hoped to have the court decide that they were not putting mine drainage into the streams. Most of the smaller companies made no attempt to keep drainage from the streams, and some months later all were cited for contempt of court. During the hearing in contempt proceedings, a combination drainage tunnel and flume for directing the

Fig. 4.—Connection between drainage flume and tunnel near Indian Creek.

mine drainage into Indian Creek below the Pennsylvania Railroad dam was decided upon as the best solution of the problem. Four companies undertook the construction and financing of the drainage project and it was completed about a year later. At present, this scheme comprises tunnels that total 4 miles in length and flumes above ground of more than 2¾ miles (Fig. 4). For the tunnel, nearly 3 miles had to be driven especially for the purpose, and the remainder was mine entries previously driven in regular mine operations. Needless to say, this work was expensive and requires more or less constant attention. The writers have been told that, as compared with continued operation of chemical treatment plants already installed, the drainage tunnel was considered the more desirable of the two schemes for disposal of acid waste, even though the court had permitted the former to be used.

With the exception of four companies who were financially able to enter this disposal scheme, most of the companies in the valley had to seal their mines or submit to having sealing done by the sheriff of Fayette County. Bills for the work were submitted to the companies later. As a result, much trouble and bitter feeling were caused by the situation.

A few of these mines so sealed have been visited at intervals by one of the writers to see whether water continued to flow therefrom, and if

so, whether the acidity and volume decreased. Of the eight so noted, sealing seems to have resulted in completely stopping the water after a period of four years from three of them, and in two others at least in the dry season. In the remainder, the volume of water seems to have decreased markedly and in those from which water issues from behind the seals, acidity has markedly decreased.

This litigation in Indian Creek Valley is thought to be the first and to date the only one of importance where a decision has been rendered against the mining companies. Not a few similar suits have been entered and after the decision described, the local press contained a number of notices of others entered or about to be entered. It was shown during court testimony that about 75,000 people were dependent on the reservoir of the railroad company for a domestic water supply, and this fact is generally admitted to have been the deciding factor in the court's order. Nevertheless, it has established a precedent of considerable importance in the problem as a whole.

Interest of Water-supply Companies

The whole subject of stream pollution is of paramount interest and importance to water companies. In thickly settled regions, the necessity of constant, close and intelligent supervision of treatment of waters for domestic use is a problem which most people do not realize. Water purification, both chemical and bacteriological, has so far advanced in recent years that in spite of enormously increased difficulties due to volume and variety of polluting materials, most of us receive, without interruption, a product well suited to our needs. The processes used in water purification are of necessity so delicately balanced that a relatively slight change in composition of many raw water supplies results in a great change in the method of treating it. At best, this requires some time and unless applied quickly, therefore, the change may result in entirely unsuitable water. Processes have been developed to care satisfactorily for the polluting materials that do not greatly vary in amount or composition. Acid wastes, however, are generally not of this class and fluctuations in their volumes and strength may cause great trouble in water purification, especially in streams near a point that normally is neutral. The increased cost of water purification is by no means an insignificant matter, but even that is secondary to the problem of satisfactorily handling fluctuations in composition of the raw water in many localities today. It may be well to add here that although acid water may be neutralized, a water of correspondingly increased hardness is the result. This water may next be softened, but if the original acidity is above a certain and by no means an impossibly high value, softening does not end the problem. The amount and nature of chemicals so used that remain in solution cause "foaming" or "priming" in boilers.

Therefore, it should not be assumed that chemical neutralization of mine water is a final solution of the problem, even if practicable financially and mechanically.

Sources and Variations of Acid Mine Water

The accepted theory of acid formation in coal mines is first an oxidation of iron pyrite, commonly known to miners as "sulfur." This results in iron sulfates, which are readily soluble in water as opposed to the original pyrite. Subsequent oxidation and hydration result finally in the formation of varying amounts of iron oxide, commonly known as "sulfur mud" and definite amounts of free sulfuric acid.

Pyrite occurs in widely varying forms and quantities in coal, also in top or bottom strata. As a rule, bottom coal and fireclay immediately beneath the coal have the highest percentages of pyrite. Not infrequently, top strata or coal also contain relatively large amounts of pyrite. These conditions seem to remain fairly constant for the different coal beds, but wide variations are to be noted in different sections of the same mine. As might be expected, therefore, the acidity of water may and usually does vary considerably in different mines in the same bed and even in different parts of the same mine.

Old, worked-out sections of a mine are almost invariably the sources of the most acid water and information from many mining men seems to indicate that pillar drawing generally results in increasing the acidity of water as noted by its color and corrosive effect on metals later. While the coal itself no doubt contributes to general acidity, it is by no means certain to be the deciding factor. In some mines crystals of iron sulfate have been observed on the face of coal *in situ* in considerable quantities, and in one in particular these crystals were said to work out of the ribs sufficiently to fill up rooms and entries. This extreme condition was indicated by piles of the crystals observed along ribs in the sections visited, from a few inches to 24 in. in depth. Analyses showed the material to be almost chemically pure ferrous and ferric sulfate in about equal proportions. On the other hand, as mentioned before, the most acid water is generally observed to be coming from sections long since worked out and here the acidity is certainly not due to the presence of coal. Likewise, in strip pits from which the coal has already been removed (and cleanly) we have observed the most acid water of a dark brown color.

Effect of High Pyritic Content

Recently, we have been studying mines known to be situated in areas having a high pyritic content. Previous work has shown that some of these give off highly acid waters. Theoretically, at least, all should be similar in this respect. Generally speaking, this has been found to be true in a preliminary survey and the effluent has usually been from two to

five times as high in acidity as is generally considered average acidity. Three of the total of 31 mines visited had water coming from them which was not acid. No doubt later and detailed information will offer an explanation for this fact.

Effect of Outside Gob Piles

Outside gob piles are believed to be an important contributing factor in the general problem of stream pollution. Almost invariably they are composed largely of the pyrite thrown out when mining or cleaning coal. They are exposed to the most favorable conditions for acid formation so far as contact with moisture and air is concerned and thus the three factors necessary in the reaction are present. As a rule, water is trickling from these refuse piles, and when pools of water can collect around them, this water is invariably highly acid. Not infrequently they are thrown across or built out into natural streams, and in such instances samples of the stream taken immediately above and below these piles often show a marked difference in acidity.

Conclusion

We have attempted to touch briefly on many phases of the general subject of mine-drainage stream pollution in so far as the Bureau of Mines work has brought the questions to light. Almost any one of these properly requires a separate paper, but if time is available and there is particular interest in some one of these various phases of the work, perhaps it will be more profitable to emphasize this in subsequent discussion.

The authors wish to express their appreciation of the cooperation of those members of the mining industry and others who have in many ways assisted in the progress of the investigations herein reported.

DISCUSSION

(S. A. Taylor presiding)

R. D. Hall, New York, N. Y.—I note that on page 140 Mr. Leitch states that several mines are alkaline in their characteristics and that the mines mentioned are all in Pennsylvania. More numerous and more striking examples doubtless could be found by studies in certain other states. In Alabama, for instance the water that leaves the coal washery does not kill fish, and often the fish in the dams which have been erected to receive this water have multiplied to such a degree they have become a nuisance. This happened with water from the washeries of the Tennessee Coal, Iron & Railroad Company. I am told that many of the mines of West Virginia have more or less alkaline water. The water there usually does not carry sulfur mud. I am disposed to believe that Pennsylvania, Illinois, Indiana and Ohio mines produce more acid water than the mines of other states.

There are two kinds of coal, designated by the geologists "paralic" and "limnetic" coals. The limnetic type is coal produced from peat which was near the sea level. Consequently the seams or the measures interstratifying them were frequently invaded by the sea. As a rule the coal contains much pyrite as a result of that invasion.

The paralic coal is the coal laid down on higher ground. In many cases it rests on and is surmounted by arkosic sediments; that is to say, by sediments which contain many of the original materials out of which this earth was built. They were the result of the erosion of unchanged or little changed igneous materials. There are such coals all along the Allegheny front.

In the anthracite region the coals are paralic in the sense that they were too far above sea level when deposited to be invaded by the sea and in a degree also in the sense that the sediments which underlay and overlay them were composed of more arkosic material than the sediments of the bituminous region.

Nevertheless there was enough sulfatic material from the Mississippian measures to give the waters of the anthracite region an acidic, usually a strongly acidic, character, the folding in many places raising the Mississippian with its marine beds above the level of the Pennsylvanian. This folding doubtless came largely after Pennsylvanian time, nevertheless it had its effect on the coal measures, the waters from eroded Mississippian beds percolating through the cracks in the shrinking peat.

However qualified may be the arkosic character of the anthracite measures, the southern part of the Appalachian uplift did afford this material, and for that reason the waters that come from the mines in that section do not exhibit the acidity manifested by the coals in the State of Pennsylvania. People have rather exaggerated the general acidity of mine waters, because they have taken their evidence exclusively from the states mentioned—Pennsylvania, Ohio, Indiana and Illinois.

I do not believe that this acidity is a serious problem in Europe. Great Britain, I understand, is fairly, if not entirely, free of it. There is—as one authority who has made many experiments asserts—a soda complex in the roof which is more active than the lime complexes which we find in the states that I have mentioned, and therefore more capable of reducing the acidity caused by the oxidation of the pyrite. Out West also the waters are not of the same character as in the states named.

G. S. Rice, Washington, D. C.—My observation is that Great Britain and the Continent do not have the problem of acid mine water which confronts mining men in some districts of the United States, for two reasons: (1) Coal measures of the Carboniferous age generally contain a large amount of interbedded shales and clays which are practically impervious to water, as in the Illinois longwall district where in some instances the mines have had less than 100 ft. of cover under streams and water-bearing strata, yet are perfectly dry; (2) in European mines the depth is much greater, so that most of the mines of the present time are deeper than 1200 ft. Usually they are overlain by shales and also surface clays or marl deposits, therefore, except in the vicinity of geologic faults, water does not enter into the mines except as it may come down from the old mines at the outcrop.

My observation has been that mines of England and Europe are dry at the faces. A large amount of water may come in through a shaft or at a fault but this has not come in contact with pyrite or sulfides and hence is not acid.

I had occasion to make a formal inquiry and found that there have been few instances in which the acidity of the mine water has been important.

A. B. Crichton, Johnstown, Pa.—I had to do with the Indian Creek litigation referred to. At that time the Pennsylvania R. R. wondered what the situation was in Great Britain and parts of Europe, and what they did there. I made a hurried investigation in Great Britain, and found that their waters were mostly alkaline, and any that were acid were very slightly acid. It was thought that this was due to the thick chalk formation overlying the coal, which greatly increased the alkalinity of the ground waters before they reached the coal.

As Mr. Rice has said, the coal seams are very deep and less of the percolating water reaches the deeper mine workings. There is not as much mine water as in the

mines of this country, and as most of the mines are near the coast, mine-drainage stream pollution would never be the problem it is in this country. Many of the mining plants use, without trouble, untreated mine water in their steam boilers.

Mr. Hall may be interested in hearing that the mine drainage of the northern West Virginia mines is highly acid, although in southern West Virginia it is less acid, due perhaps to the lower sulfur coals.

Some years ago I thought that mine-drainage stream pollution was one of the most serious problems confronting the coal industry. But since we have been discussing the economic situation through which we have lived during the past eight years or so, I would say now that it is next to the most important question, or perhaps the third most serious problem. But the thing that concerns me is that no satisfactory solution of the problem has been found.

The attempt to treat the mine water with lime in Indian Creek Valley caused so much sludge that it was impossible, or almost impossible, to handle it. It is a slimy yellow mud, too thin to shovel and too thick to pump. I think the reason the tunnel was finally decided upon as the most feasible plan for keeping the mine water out of Indian Creek was that the lime treatment was not satisfactory in method or results. This treatment was no cure, as it made a very hard water, and after this expense it was yet unfit for any use. When water contains 15 grains per gallon of acid sulfates it cannot be treated satisfactorily, and usually there is a comparatively short time between the time it begins to need treatment and the time at which it becomes unfit for treatment.

It becomes a serious problem for Pittsburgh and other cities along the Allegheny and Monongahela rivers. With increasing quantities of mine drainage being dumped into the rivers and their tributaries, because of the increasing coal development, their future water supply for domestic and industrial needs seems in danger. This year, the river waters have been acid a large part of the summer, seriously affecting river transportation. The river water has been unfit for boiler use, and is attacking the boats and barges.

Sealing abandoned mines, which we did find fairly satisfactory in Indian Creek, has been referred to here. Some of the mines that went to the dip from the openings were not so hard to seal up, and in that event only the main openings were sealed at outlets. In a mine at the upper end of Indian Creek Valley, where the coal measures went to the rise about 8 or 9 per cent. from the drift mouth, and where sealing could not be expected to shut off the water, we closed all openings around the crop, sealing off the air with fairly satisfactory results, as the acidity has been considerably reduced. The fish in Indian Creek at one time were killed for several miles below the opening, but that stream is clear now. The same thing is true in most of the other mines sealed,—the acidity has been reduced and the quantity of water flowing from the mines has been lessened, in some cases stopped entirely.

R. D. Leitch.—A short distance north of Pittsburgh, in the Thick Freeport area, out of 17 mines investigated, 6 were found not to have any acid water, 6 had both acid and alkaline waters, and of those having acid water, only one was strongly so. A partial explanation of these facts, without doubt, is that the water coming into the mines in the Thick Freeport bed is of an unusually high alkalinity. I found many samples ranging from 800 to 1200 parts per million. A normal, pure surface water, taking Indian Creek as an example, generally runs from 17 to 19 parts per million. That gives an idea of what surface waters are. There are waters coming into the Thick Freeport area having an alkalinity of at least 810 parts per million, and they are able, therefore, to neutralize considerable amounts of acid when formed or to prevent such formation. At the same time the Thick Freeport bed is low in total sulfur and considering that about 0.6 per cent. of it is organic sulfur, which cannot

enter into an acid formation, there is not much pyrite left to form acid. But of course the highly alkaline water coming in is the explanation.

R. D. Hall.—What is the character of that alkali?

R. D. Leitch.—The water is a so-called "bicarbonate" water; that is, it contains relatively large amounts of sodium and potassium bicarbonates, and carbonates as well.

A. B. Crichton.—I would like to add a word to what Mr. Leitch has said about the tunnel and the gob piles in Indian Creek. Even after taking that half million or million gallons of mine drainage out of Indian Creek and sealing up the mines to which reference was made, the stream water above the dam was hardly different from its condition before treatment was given, and in my opinion this can only be attributed to the natural increase of water from these gob piles in recent developments. There is no doubt that the gob piles caused a great deal of acid in the streams.

C. W. Gibbs, Pittsburgh, Pa.—I think we must be operating one of the mines that had both acid and alkaline waters. Two or three years ago we were shut down from active operation for about a year and a half during reconstruction of the tipple and bottom. Prior to that time we could put in any kind of pipe and it would last indefinitely, but since the shutdown we have had all sorts of trouble with acid condition, especially in one section.

R. D. Leitch.—It is probable that the water in those sections had an opportunity to come in contact with air, which might have resulted in an increase in free acid content.

C. M. Lingle, Nemacolin, Pa.—Have any of your investigations been of coal seams above which is a thin seam of low-grade iron ore?

R. D. Leitch.—I have had no experience of that kind. In fact, I do not know where to go to find mines of that kind. I think such deposits of iron would not determine whether the drainage, or even entering waters, would be acid or alkaline.

R. D. Hall.—The water in the Van Lear mine of eastern Kentucky was relatively alkaline, yet there the roof contains large quantities of carbonate of iron. Though there are exceptions not far from the line, the mines along the Appalachian front, south of Mason and Dixon's line, rarely have any trouble from acid water. Most of them use steel pipe exclusively. Probably the presence of carbonate of iron makes little difference. It can be found in the B seam of Pennsylvania, where the worst kind of acidic water is produced.

R. D. Leitch.—Iron is found in the form of iron carbonate or bicarbonate. If it is not present in connection with coal deposits in the form of pyrite, the sulfates of iron are not formed and subsequently sulfuric acid, although the yellow precipitate of iron oxide often is present. This often leads observers to believe that the waters from which it has been deposited are acid, but this is not always true.

The Roof of the Pittsburgh Coal Bed in Northern West Virginia

By Lee M. Morris,* Morgantown, W. Va.

(Fairmont Meeting, March, 1931)

The Pittsburgh bed, lying at the base of the Monongahela series, is probably the most famous bituminous coal bed in the world; famous not only for the product yielded in mining, but also as a key horizon in the search for oil and gas. It is the most important coal bed in northern West Virginia from the standpoint of extent, thickness and regularity. The general section is shown in Fig. 1.

Roof Strata

The strata that superimpose any coal bed vary more than the coal itself, and the Pittsburgh bed is an excellent example of this fact. Obviously, there were several changes during the period of deposition of this overlying material. Locally, sandstone occurs above and in contact with the coal. Sometimes a shale, or a clayey shale, rests on the bed and is often overlain by sandstone. Throughout the greater portion of northern West Virginia, however, the basal member of the immediate roof is a black or gray clay, which is usually slickensided. This material is called "draw-slate" by the miner. The color is due to the carbonaceous matter and the slickenside is the result of pressure, or movement, which may have come during or subsequent to its consolidation. This part of the roof requires especial consideration, because it is not self-supporting after the coal has been removed from beneath. The roof coal, 8 to 24 in. thick, is left directly below the draw-slate to support it and also to protect the clay from changes in temperature and moisture in the air, thereby preventing disintegration. The clay or draw-slate quickly changes its physical character when exposed, and falls if it is not adequately supported. Timbers alone are not effective in holding up this material, because it softens, crumbles and falls around the supports. Timbers hold up only the material that is directly over the posts and crossbars.

In Monongalia County

In the northeastern part of Monongalia County, between the Monongahela and Cheat Rivers, sandstone is usually found immediately above

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and in contact with the coal bed. Draw-slate, which seldom is more than 2 ft. thick, sometimes lies between the coal and sandstone.

The immediate roof in the northern part of Monongalia County, west of the Monongahela River (Cass district and the northern part of Grant district), is composed of clays, rider coals and shales (Fig. 2). This interstratified material, having a total average thickness of about 7 ft. constitutes a soft roof which breaks easily. A black or gray slickensided clay, 4 to 36 in. thick, rests immediately upon the coal. This stratum, averaging 18 in. in thickness, pinches and swells continuously; it softens, expands and falls a short time after exposure. Immediately overlying the clay is 9 to 18 in. of hard, bright coal which breaks into large rectangular blocks. Its thickness is more regular than that of

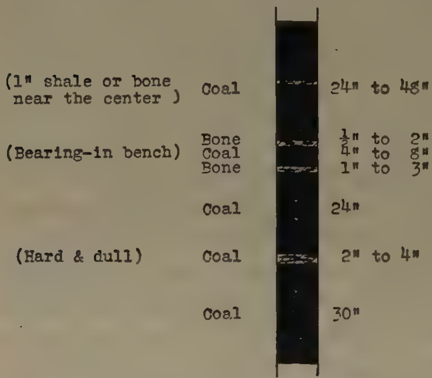

FIG. 1.—GENERAL SECTION OF PITTSBURGH COAL IN NORTHERN WEST VIRGINIA.

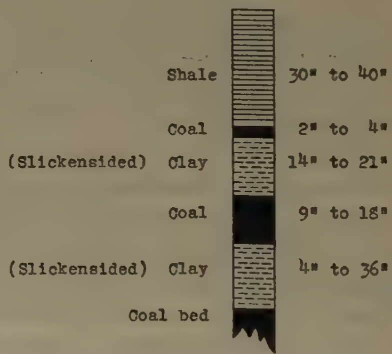

FIG. 2.—SECTION OF IMMEDIATE ROOF STRATA IN CASS DISTRICT, MONONGALIA COUNTY.

any other member in the roof division, averaging 1 ft. On Scotts Run it contains 6 per cent. sulfur, 18.37 per cent. ash, 36.41 per cent. volatile matter, 0.66 per cent. moisture, and 44.56 per cent. fixed carbon. It stays up longer on exposure than the clay directly below. Another clay seam which is similar to the lower one in color, structure and thickness occurs above the coal. These two slickensided clays are irregular in thickness, and detailed measurements made a few yards apart may show extremely dissimilar sections.

Another rider coal, about 4 in. thick (maximum), lies above the upper clay. Southward, this seam is the first member in the roof division to thin and eventually it disappears entirely. A fine-grained, thinly laminated shale lies immediately above the coal; or above the clay if coal is not present. Thin streaks or lenses of coal are frequently found in this stratum. This shale is very hard and breaks into almost perfect cubes. Apparently it is not affected by weathering when exposed to the mine air. It does not fracture and fall except along pillar lines.

Southward to the vicinity of National, in the southern part of Monongalia County, these members have thinned and the top ones have disappeared, but the same slickensided clay immediately overlies the greater portion of the coal bed and maintains an average thickness of about 1 ft. A coarse, white micaceous sandstone, 2 to 20 ft. thick, rests upon the clay. Sandstone locally rests upon the coal. A fine-grained brown sandstone, ranging in thickness from 1 to 4 ft., occasionally lies between the clay and the coarse white sandstone; a dark gray shale, generally 1 to 4 ft. thick, overlies the coarse sandstone. It exceeds 15 ft. in thickness where the sandstone is thin, but it is not persistent. The rider coals have disappeared in this locality.

In Marion County

Along the Marion-Monongalia County line the average roof structure is somewhat similar to that in the northern part of Monongalia County but there are considerable variations of local extent. The slickensided clay, 2 in. to more than 40 in. thick, generally immediately overlies the coal. It averages less than 1 ft., but its thickness is continually changing. Here the rider coal has reappeared, but is not very persistent. Its usual range in thickness is from 3 to 11 in. Sometimes a slickensided carbonaceous shale lies at its horizon. A stratum of clay, or clayey shale, 20 to 40 in. thick, which is darker in color but not as slickensided as the lower clay, overlies this thin coal. The upper rider coal, lying near the center of the top clay, was observed at one place where it measured 10 in. A sandstone overlies the draw-slate.

Locally, the structure deviates from these general characteristics. At points where there is a change in character and kind of interstratified material, there is no consistency in the composition of the various strata. In a limited area the roof is composed entirely of slickensided clay several feet thick, which falls in large wedge-shaped pieces. Where this condition exists it is practically impossible to support the roof. A few hundred feet away the immediate strata may have changed to sandstone and sandy shale. One such section, in ascending order, consists of the following materials: a white sandstone, 8 in. thick, containing layers of coal in the bedding planes, usually ¼ in. thick; a 15-in. stratum of gray sandstone with a little coal and shale in the bedding planes; overlying this is 26 in. of micaceous shaly sandstone, containing pockets of small pebbles as well as numerous layers of coal; above this is 24 in. of finely laminated shale containing alternate black and white bands and over this is 7 in. of black shale, streaked with coal. Gray and white shales, some of which are coarse, sandy and micaceous, rest upon these materials.

In the northeastern part of Marion County, along Paw Paw Creek, the roof has a more uniform structure. The characteristic slickensided clay, 6 to 10 in. thick, rests upon the coal. The lower rider coal, ranging

from 1 to 14 in. in thickness, and averaging about 11 in., is persistent through this area. Between the face cleats of this coal many layers of calcite crystals were noticed. Locally, a nonpersistent canneloid shale, 3 to 4 in. thick, rests immediately upon this coal. This layer of shale probably is the result of the raising of the peat bog, which disturbed the coal and permitted the carbonaceous matter to intermix with the mud. Approximately 4 in. of gray shale lies above the thin coal or canneloid shale where present. A micaceous shaly sandstone lies above the laminations of coal and shale. Small, as well as large plant fossils are abundant in this material. The thickness of this rock is not known; in some roof cavities more than 11 ft. are exposed.

In the southern part of Marion County (Lincoln district) the slickensided clay, which generally rests upon the coal bed, has a probable average thickness of 8 in. but varies from 2 to 24 in.; sometimes a hard black shale containing specks of carbonaceous material and small plant imprints is in contact with the coal bed. The rider coal, having an average thickness of 7 in., is not always present; its maximum thickness is probably 1 ft. Occasionally, the second rider coal is present a few inches above the lower one. A gray shale, or a clayey shale, about 1 ft. thick lies above the coal and draw-slate. Above the stratum just described is a jointed, hard, black shale, approximately 18 in. thick, which on exposure to the air deteriorates rapidly and cracks along its pronounced vertical joints and falls. Above this material is a dark, smooth shale which breaks only along pillar lines.

In Harrison County

Along Mudlick Run, in northeastern Harrison County, the immediate roof is made up of the characteristic clay, 0 to 2 ft. thick. It appears to consist of two formations, the lower few inches being more susceptible to moisture, thereby deteriorating more rapidly. Over this stratum is sometimes a hard, dark gray shale, approximately 3 ft. thick, which has very pronounced vertical cleats. A thin bed of coal, ranging in thickness from 3 to 7 in., is sometimes found resting on the draw-slate. Although this coal is not persistent, traces may be found in many places. It has been so badly disturbed locally that it appears to be a part of the draw-slate. Above the strata just described the roof is made up principally of sandy shales and sandstone. In some places, sandstone lies directly upon the coal and where it is separated from the coal possibly only a few inches of draw-slate intervene. The roof in this vicinity is exceptionally strong, breaking only along the pillar lines.

In the vicinity of Wolf Summit, in the southeastern part of Harrison County, the strata overlying the coal are somewhat different from those generally encountered. The rider coals no longer are present. At times a thin carbonaceous shale is encountered at its horizon. With this

one exception the roof consists of a persistent, light gray fireclay shale of which the aggregate thickness is reported to be 15 to 20 ft. Although this material is comparatively hard when it is first exposed, it soon becomes soft, disintegrates rapidly and is a little plastic when moist. Approximately 4 in. of impure coal adjacent to the roof is left in place to protect and prevent this roof from falling. Under normal conditions this is ample to hold the roof, but when considerable water is encountered this material expands, thereby breaking down the coal and making it necessary to timber heavily.

In the extreme southern part of Harrison County, along Browns Creek, the immediate interstratified material consists of clay, clayey shale, and locally sandstone and dark shale containing a high percentage of coaly material and plant remains. An average section in ascending order is as follows: 1 ft. of clay, then 10 to 48 in. of a pyritic clay which is slightly slickensided, above which there is possibly 20 ft. of clayey shale, streaked with coal and somewhat slickensided. Locally, several feet of carbonaceous shale rests upon the coal bed.

Roof Support

Inasmuch as there are various kinds of roof strata and mining methods, one would hardly expect to find the same standard system of roof support in general use throughout this territory. Some similarity exists, however, where the importance of roof control is not only realized, but practiced. All roof should be considered a potential source of danger and every precaution should be taken to give adequate protection to those exposed; this is especially true where top coal is left. The roof coal itself has very little supporting strength and it is apparent that the immediate roof of the Pittsburgh coal bed throughout northern West Virginia has little or no self-supporting ability. Weak spots in the overlying material are hidden by the top coal, and unless a uniform system of timbering is strictly adhered to, regardless of appearance of the roof, falls resulting in accidents, delay and increase in production costs are the final results. It is unwise, therefore, to depend on the top coal as an artificial means of supporting the overlying strata because it gives one an impression of false security.

The roof coal acts as a uniformly loaded beam fixed at both ends; in the center of the room or entry, where the greatest tensile strain occurs in the top coal, it cracks along the bottom face, and on the sides, where the maximum compression is centered, it crumbles or crushes just as coal crumbles when subjected to compression tests in the laboratory. The length of the roof span is a factor in the relief of this condition, because in a narrow place the result is shearing of the roof coal along the rib, whereas in a wide place there is merely deflection or bending. The shearing of the roof coal along the rib is also often the result of its being

weakened or shattered when the other coal is shot down. When the coal is top-cut the roof coal is protected from the blasts. Because of these facts, timbers are necessary to support adequately the roof coal, which is simply a good covering to protect the overlying strata from the mine air.

Where systematic timbering is followed, the method consists of a straight row of posts, not more than 5 ft. apart, extending within 8 ft. of the coal face and approximately 2 ft. from the side of the track to allow for clearance. In some mines it is customary to set a row of posts on both sides of the track. Safety posts are in common use throughout this territory. As a rule, they are set close to the coal face, at or near the center of the working place. They should be used in every working place, inasmuch as many of the accidents from falls in the Pittsburgh coal bed are caused by the falling of unsupported pieces at or near the face.

As a rule, very little entry timbering is used unless the roof shows signs of breaking. Timbering along entries consists either of posts set along one side of the track or of headers. The crossbar is supported either by two full-length posts which rest on the bottom or by two short leg posts resting on the rib coal. When steel is used for crossbars, one end is sometimes hitched in the rib and the other end set on a short post or block placed also in a hitch cut in the rib. In mines where the roof is wet and very soft, like those in the vicinity of Wolf Summit, the method of support along the main haulageway consists of plank lagging supported by steel crossbars hitched in the rib.

Conclusions

The writer concludes that as long as men work under a hazardous roof such as overlies the Pittsburgh coal when it is inadequately supported, fatal underground accidents will never be materially reduced. Accident prevention in coal mining is not a simple problem, but not until the importance of systematic timbering is fully recognized and practiced will accident prevention progress.

DISCUSSION

(*Thomas G. Fear presiding*)

T. G. Fear, Fairmont, W. Va.—Mr. Morris, as you study this Pittsburgh coal do you find any connection between character of roof and analyses?

L. M. Morris.—I have never considered that and I do not know that anyone else has.

T. G. Fear.—The reason I ask is that where the sandstone is directly above the C prime coal in western Pennsylvania the sulfur content materially increases.

J. D. Sisler, Morgantown, W. Va.—The normal stratum above the Pittsburgh coal is a mixture of shale, slate and thin layers of coal in local areas. The sandstone that normally lies above this intermixture is in direct contact with the uppermost of these shales, which normally lie above the Pittsburgh coal. On the old surface on

which these shales were deposited by streams, in some places so much of the material was removed, the stream came all the way to the top of the Pittsburgh coal and its channel was refilled by clay, sand and gravel. Some geologists believe that the sulfur content of a coal bed has been reduced by the action of roots of the trees that grew in the swamp. Another idea is that there has been a bacterial action on the base on which these plants grew and that this action may have reduced the sulfur content of the coal bed where the stream had eaten entirely through and exposed the Pittsburgh coal bed.

Mr. Morris' paper is unique. In all the years of mining Pittsburgh coal there never has been a paper written about the roof of the Pittsburgh coal, on a geological basis. The paper was written at my suggestion, and it is my hope that geologists will make a study of the roof of the Pittsburgh coal in Western Pennsylvania.

Last year, Mr. Lambie and I represented the West Virginia operators before the Interstate Commerce Commission. The question of roof over the Pittsburgh coal and its effect upon the recovery of coal and increase of costs came up and we searched the literature but found nothing we could use as testimony.

J. R. Campbell, Pittsburgh, Pa.—I am much interested in the question of the effect of roof conditions on the chemical qualities of the coal; particularly sulfur. My observation over a good many years has been that with a good roof the chemical qualities are better.

Mr. Sisler is credited with the statement that "No other bed approaches it in area and in persistency of physical and chemical character, and commercial possibilities, with the possible exception of the No. 6 bed in Illinois." I can easily subscribe to two things in that statement but I would like to have Mr. Sisler tell us what he means when he speaks of "chemical character" of the Pittsburgh seam of coal, which we know does vary in different districts, especially in the matter of sulfur content and in its distillation products.

J. D. Sisler.—There are few coal beds that do not have large local areas wherein the sulfur and ash contents vary. Consider the Pittsburgh coal bed where it can be mined and cleaned at a profit: in all of that area the Pittsburgh coal is minable, with the exception of the southern part, particularly south of Harrison County, West Virginia, into Charleston. If all of the variations in sulfur and ash content were averaged, I doubt whether they would vary more than 6 to 8 per cent. It is a characteristic that is not found in any other coal bed in the United States, with the exception of Illinois No. 6; the chemical qualities are quite consistent.

J. R. Campbell.—The No. 6 Illinois bed is more persistent in chemical quality than the Pittsburgh bed. Pittsburgh coal is queer in this respect, because the other characteristics are so similar.

T. G. Fear.—Mr. Campbell, we are mining some Pittsburgh coal in this region that is variable in regard to sulfur. We are forced to take samples every 100 ft. Often we will have 12 or 15 rooms working in one panel and suddenly have to stop and take samples every 50 ft. We have to watch this carefully.

J. W. Paul, Pittsburgh, Pa.—Mr. Morris' paper is a valuable contribution to literature in respect to the Pittsburgh coal and its roof. While Mr. Morris has made a study largely on the geological basis, our study is largely on the physical basis, in regard to the character of the roof. I feel that what he has contributed here will be a great help in our study and in studies made by other persons on the same subject.

Shaker-chute Mining, Northern Anthracite Field

By Kenneth A. Lambert,* Scranton, Pa.

(Pittsburgh Meeting,† September, 1930)

In the Anthracite Region there are considerable areas where the original mining was done 50 or more years ago. In this original mining, the pillars that were left proved inadequate in size for the support of the roof, which has since collapsed as a result of squeezes, and completely filled the original openings. In an endeavor to recover the coal which remains in the pillars in such areas, the reopening cost for development, as well as for pillar recovery, has been excessive in many instances. Various methods have been tried out with a view to lowering the cost, in order that the greatest possible percentage of recovery of the reserve tonnage may be realized economically.

Ordinary hand mining necessitates the driving of development road 10 ft. wide in the clear between timber legs, with 6 ft. of clearance over the rail to accommodate mine cars; collars or crossbars on the timber sets have been difficult to maintain in these openings in the heavily caved condition, especially in beds where the overburden is great. Numerous failures have occurred on these development road timbers because of roof movement resulting from removal of pillars. The actual removal of pillars from this development work has been costly because full-width openings for transporting mine cars to the face have to be maintained by cleaning caved material from alongside the pillars, or skipping up along the pillars where pillars of sufficient width are found.

Shaker-chute Methods

A method of developing such caved conditions and recovering pillars tributary to this development, utilizing shaker chutes, is being tried out with considerable success at Loree colliery. An outline of the chute and apparatus is shown in Fig. 1, which indicates the location of pillars and caved material. The cost of both labor and material required in the recovery of such coal is considerably less than by ordinary hand methods.

In order to try out this method of mining a location was chosen where a tightly caved area had been developed by ordinary hand methods to within 300 ft. of the property line. The original bed height was 12 ft., with an overburden of 325 ft., but on account of the squeezes which

* Superintendent, Loree Colliery, Hudson Coal Co.

have occurred since the original mining was done the bed has been crushed to a height of 10 ft. in the remaining pillars. The original chambers will average 24 ft. in width and the remaining pillars 18 feet.

Development openings in this 300-ft. block were set on lines that will provide a slight grade in favor of the material coming from the face. The development opening was driven to accommodate small timber sets, collars being 4 ft. between notches and the legs at the bottom 6 ft. apart with a clearance over the bottom from 5½ to 6 ft. A heavy type of shaker conveyor was installed for the handling of material from the face to the mine car placed on the track in the old hand-developed counter, the chute being suspended from the timbers so that four cars could be placed by the electric locomotive for loading.

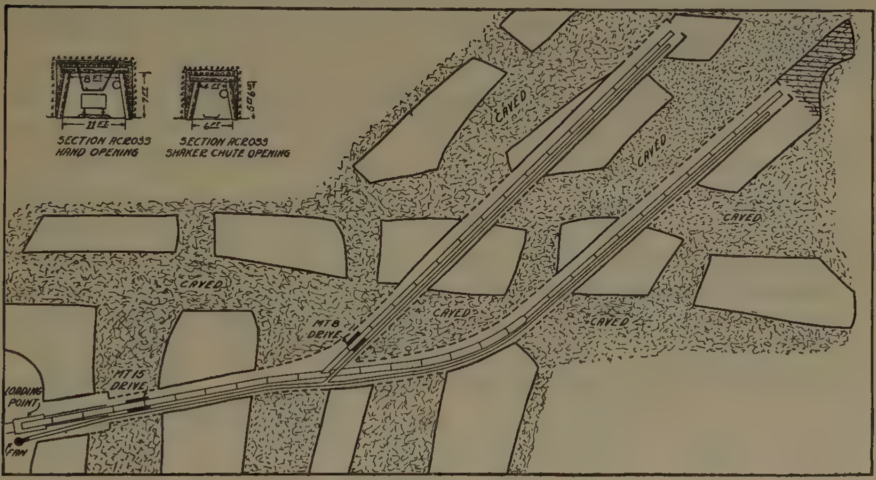

FIG. 1.—SHAKER-CHUTE MINING, HUDSON COAL CO.

Development was extended the 300 ft. to the pillar along the property line and a swivel connection was then turned off the main chute up into the last pillar. A light type of conveyor was installed to handle the coal in the second pillar from the face delivering to the main chute.

In recovering the pillar coal, openings to accommodate the small timber sets are driven within the pillar, the crushed coal on the side and top being held with lagging. Every effort is made to keep the size of the opening as small as possible, both on development and driving up through the pillars, so that the probability of a squeeze in these openings is slight. The shaker chute is carried on one side of the development opening or chamber opening, so that there is sufficient passage room for men traveling in and out and for the handling of material. The timber material is handled on a small truck, which has been built to run on the shaker chute, pushed back and forth by hand.

Ventilation of the working places is accomplished by means of a small blower fan, powered with 440-volt, alternating current, 5-hp. motor through 12-in. Ventube, a Y being provided in the Ventube to split the ventilation between working places.

The heavy shaker chute is 200 ft. long and an extension of 170 ft. has been placed on the end of this main chute for the operation of the inside places. The light shaker chute is operated with 250 ft. of chute.

Two men are employed in each working place (one miner and one laborer) and one man is employed at the delivery end of the main shaker chute to attend not only to the proper trimming of the cars but to the manipulation of electrical switches, which are centrally located for the operation of both shaker chute and blower fan.

Comparative costs of the ordinary hand mining method in development and coal recovery in this condition and of the shaker chute method are given in Table 1.

TABLE 1.—*Costs of Recovery of Coal in Crushed Area*

	Hand Method		Shaker Chute	
	Labor	Material[a]	Labor	Material[a]
Cost per yard	$62.80	$3.25	$17.49	$1.67
Cost per yard, development:				
Caved chambers	60.23	3.16	14.67	1.64
Crossing pillars	65.47	3.94	14.39	1.79
Cost per yard, driving through pillars			15.98	2.39
Cost per ton, on total recovery	7.40	0.40	2.07	0.20
Cost per ton, labor and material	$7.80		$2.27	

[a] The material cost includes maintenance and depreciation charges against the shaking-chute equipment.

This comparison shows distinctly the advantages of the shaker-chute method of recovery and these are even more strikingly shown from the cost which obtained during the months of May and June, after the men became more familiar with this method of mining; the cost for labor and material for these two months were $1.93 per ton for May and $1.75 per ton for June.

While the length of shaking chute that can be driven satisfactorily from one unit may be limited to possibly 300 to 400 ft., it is believed that this method of recovery could be made to fit physical requirements by having shaking conveyors deliver one to the other, so that the length of development would not be limited to the operating length of one shaker conveyor unit.

Mechanical Mining

By Eugene McAuliffe,* Omaha, Nebr.

(Pittsburgh Meeting, September, 1930)

The term "mechanical mining" carries an ambiguity which justifies a preliminary word of explanation. All mining activity conducted in this day is more or less mechanical; that is to say, power expressed in the form of electricity, steam or compressed air is used in some portion of practically every mining activity of consequence. "Mechanical loading" is more nearly correct than "mechanical mining" in reference to present practice.

The first substantial effort made in the direction of mechanical mining, subsequent to the application of steam for hoisting and underground pumping purposes, was that of substituting for hand labor undercutting and shearing by power-driven machines. The undercutting of coal by hand represented at one time the most arduous task confronting the miner. Occupying a kneeling position in the thicker beds, or lying on his side in the thinner coal, the task was tedious and exhausting.

The first attempt to substitute crude air-driven, and later electrically driven, "punchers," as the percussion type of machine was called, met with much the same opposition on the part of the workers as was expressed by workers in the other industries when any innovation, whether labor-saving or otherwise, was introduced. We should not be too critical of this attitude of mind shown by the mine workers. The peculiarly isolated character of the work carried on in semidarkness by a class of men who to a large extent lived and held themselves aloof from the other branches of society led long ago to the development of a craft consciousness which they have found difficult to surrender.

It was not until 1891 that a definite record of the results obtained by the use of undercutting machines of all types was made available, the quantity undercut in that year totaling 6,211,732 tons of bituminous coal from all American mines. The volume of bituminous coal undercut by machines in 1928 totaled 369,687,007 tons, or 73.8 per cent. of the nation's production. In the light of this experience, we can properly feel that the work of introducing the coal-loading machine as a substitute for hand shoveling is making substantial progress.

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* President, The Union Pacific Coal Co.

The quantity loaded mechanically in bituminous coal mines, as shown by the figures published by the U. S. Bureau of Mines commencing with 1923, including advance figures for 1929, are as follows:

Year	Tons	Ratio, Per Cent.	Year	Tons	Ratio, Per Cent
1923	1,880,000	100	1927	14,559,000	733
1924	3,496,000	186	1928	21,559,000	1146
1925	6,148,000	327	1929	37,862,000	2014
1926	10,022,000	533	1930	48,824,000	2597

The coal loaded mechanically for 1930 suggests an increase of 23.7 per cent. above that of the preceding year, the figures shown covering underground loading devices only; some 22,000,000 tons of bituminous coal and lignite were loaded by power-driven shovels in surface strip pits in the year 1929. Paralleling the increased application of undercutting machines, it is worth noting that electric locomotives are now in service in American coal mines producing 87 per cent. of the nation's coal.

A British Example of Machine Mining

That "mechanical mining of coal" is at the best a relative term was brought forcibly to my attention on the occasion of the ninety-first meeting of the Institution of Mining Engineers, held at Birmingham, England, July, 1930, where a most illuminating and informative paper was read on the introduction of machine mining at Newdigate colliery.[1] The author of this paper presented his experience under what was referred to as "the complete mechanization of the coal-getting operations at Newdigate colliery." The mechanization of this mine, however, while substantially in advance of the majority of British coal-mining operations, has not been extended beyond the usual point, except as to the introduction of the undercutting machine and face conveyors of the shaking and belt types, supplemented by gate end loaders and gate conveyors, the last used to transport coal from the face transport units into the pit cars. The electric locomotive is seldom used in British mines, but various types of rope haulage, which are properly termed "mechanical haulage," are employed. The Newdigate operation does not use machinery for lifting the coal from the floor of the working face to the belt or conveyor; this work is done by men using picks and shovels.

Without any desire to dwell on the British situation presented by Mr. Newey, the author feels that his results, obtained under a progressive translation from hand-mining to a process which included undercutting by machines and transport from the face to the haulage entry by belt and shaker conveyors, will prove interesting to American coal-mine managers. (See Table 1.) The figures shown by Mr. Newey are indicative of the results obtained from what we would refer to as a "partial mechanization," but which probably represented a relatively greater measure of forward

[1] D. S. Newey: Introduction of Machine Mining at Newdigate Colliery. *Trans. Inst. Min. Engrs.* (1930) **79,** 372.

TABLE 1.—COMPARISON OF DATA ON HAND AND MACHINE MINING[a]

Period	Average Weekly Output, Tons	Output per Man per Shift Worked, Tons			Length of Face Working, Yd.			Percentage of Machine-mined Face to Total	Output per Yard of Face, Tons per Week
		At Coal Face	Total Underground	Total Employed	Hand Mined	Machine Mined	Total		
June, 1927	4700	2.39	0.74	0.57	1650	...	1650	0	2.84
December, 1927	5600	2.93	0.84	0.65	1440	160	1600	10	3.50
June, 1928[b]	4000	4.21	1.05	0.80	880	380	1260	30	3.17
December, 1928	5500	5.23	1.26	0.95	250	750	1000	75	5.50
June, 1929	5700	5.59	1.40	1.02		850	850	100	6.70
December, 1929[c]	6700	5.46	1.48	1.09		850	850	100	7.88
March, 1930[d]	7600	6.20	1.67	1.23		900	900	100	8.44

[a] From paper by D. S. Newey. Reference of footnote 1.
[b] Working one shift only.
[c] Haulages unable to deal with increased output.
[d] Haulage arrangements improved and gate-road conveyors installed.

progress for this particular mine than would result from the mere application of loading machines in one of our American mines, previously equipped with undercutting machines and electric haulage. The author does not consider the results obtained by Mr. Newey as properly determinative of the results that might be obtained by the further application of machinery in either the average British or American mine, but does hold that the application of loading machines and loading methods properly selected, thereafter properly carried on, will result in substantial reductions in mine costs in many American mines.

AMERICAN EXPERIENCE WITH MECHANICAL LOADING

Many American coal-mine managers have looked forward to the introduction of mechanical loading machinery as the means whereby material savings could be made by changing the basis of wage payments, from a contract or tonnage price to a day-wage basis. At many mines, the results obtained have been disappointing, for various reasons. To begin with, it is proper to say that the contract miner was not always given due credit for the time spent in drilling, shooting and timbering his working place, and in too many cases the cost of the explosives used by him were underestimated. With these collateral activities and the cost of explosives shifted to "other costs," and thereafter with the actual tonnage "loading labor allowance" properly allocated, the money received for the loading process was at times not excessive. Mr. Newey did not hesitate to attribute much of the success gained by the introduction of undercutting machines and conveyors to the application of a tonnage price paid for each of the various processes employed, the tonnage payment basis when properly determined furnishing a definite incentive toward production.

In the conduct of our own mechanical loading operations, we had no preconceived ideas of extraordinary gains to be obtained by changing from the tonnage to a day-wage basis. We did, however, feel that the tonnage rates paid to machine runners and, to a lesser extent, to miners and loaders, were excessive. We further held to the opinion that the relatively high tonnage rates so paid were largely the result of past indifferent transportation service rendered, plus lack of organized delivery of mine timber, rails, and other materials, the mine manager in the past undertaking to keep his costs down by maintaining the minimum day force, the minimum measure of mine maintenance, and so forth. Gradually, and with each new wage contract, the miner, loader and machine runner demanded and received allowances in the form of increased tonnage rates to compensate him for the disadvantages he suffered in the form of a "slow turn," lack of timber, rails and low voltage, the last impairing the undercutting and haulage processes. Whatever other advantages may accrue from the introduction of the mechanical loader, we cannot overlook the fact that the mine manager has learned that successful and economical operation cannot be carried on unless every individual part of the process is maintained at an equally high standard.

Experience of The Union Pacific Coal Company

As a result of careful time studies made at our properties, we decided, some months ago, that the universal application of the day-wage basis payment is, to say the least, a mixed blessing. With the view of bringing up the unit production and unit earnings, we are at present employing with one type of loading machine a premium basis from which substantial results have accrued to both the worker and the company. In the application of the premium, no attempt was made to depart from the established contract scale price. On the other hand, the premium allowance is based on an extra payment credited to each individual machine that earns it and thereafter is divided equally among the men who handle the machine. This premium allowance takes effect when the average man-shift production for the crew, over a period of two weeks, exceeds a given tonnage per man-shift. If the tonnage on any particular machine for the two weeks falls below the established basis, no penalty attaches; on the other hand, the mine management undertakes to find out the nature of the disabilities under which the crew has labored, the remedy frequently resting wholly with the management. For the three months ended July 31 last, with this arrangement under way, a premium of 76.9¢ per man-shift was earned by the men serving 34 per cent. of the total man-shifts and producing 44 per cent. of the tonnage, the amount of premium paid when extended over the total number of man-shifts employed equaling 26.6¢ per man-shift. The men who received a premium

produced 50 per cent. more coal per man-shift than did those who were not premium earners.

Arguments for Increased Mechanization

What are the arguments for increased mechanization? Perhaps one of the most valid is that covered by the title of a concert hall song much in vogue in recent years, "Everybody's Doing It." There is not an industrial process, including the two greatest of all American activities, agriculture and transportation, that has not made extraordinary strides toward increased mechanization in the past 10 years. The changing processes are too numerous to mention, as they include the application of power-driven machines for the making and finishing of nearly every manufactured product, supplemented by the introduction of units of larger capacity employed in lieu of the smaller units used previously.

The U. S. Bureau of Mines, in one of its many informative reports,[2] sets forth the extraordinary progress between the year 1913, preceding the Great War, and 1929, in the use of crude oil, natural gas and water power, as compared with bituminous and anthracite coal. From this statement we gather that while coal contributes 87.1 per cent. of the total British thermal units derivable from mineral fuels in 1913, the ratio had fallen to 65.3 per cent. in 1929. In the meantime, the total B.t.u. value of foreign and domestic oil and natural gas used increased from 12.9 to 34.7 per cent. Water power, which is treated as a separate item, shows a growth from 3.4 to 7.9 per cent. during the same period.

It is only fair to ask, what is the relative measure of improvement from the processes employed in the production of crude oil and natural gas between the years 1913 and 1929, and those that govern in the production of coal? These represent a situation which is not definitely measurable, but I am satisfied that if the same relative engineering genius that has made it possible to more than double the practical depth to which an oil or gas well may be driven, and to bring about the improvements that have been made in oil-line and gas-line installation within the past 10 years, had been employed in the conduct of our mining operations, the spread in consumption between coal and liquid mineral fuels would show a somewhat different result.

There are numerous other reasons why the further mechanization of our American coal mines should be brought about, but the subject is one which is continually under discussion and which is receiving the best thought of more competent men. At the risk of "adventuring," I will suggest certain lines that are deserving of study and consideration:

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[2] Per cent. Total B.t.u. Equivalent Contributed by the Several Mineral Fuels of the United States.

1. Double and triple shifting of fewer mines would so reduce capital charges and operating costs as to set up a further measure of selectivity, tending to eliminate the superfluous mines, making new development an undertaking for more thorough consideration on the part of prospective coal mine investors.

2. Development of a system of individual machine organization underground, whereby a certain crew would be assigned to a given machine, one of the crew to be paid a wage somewhat above the average and held responsible for his machine and crew, the measure of success or failure to be determined by the results obtained, the question of safety to be given equal weight with production. The crew should be paid a stated wage, plus a premium for each ton loaded over an agreed tonnage per man-shift worked, computed over the two weeks' pay roll period, and the premium paid should as nearly as possible be predicated on a fifty-fifty division of the reduction in unit labor costs achieved.

3. If we are not to continue to lag behind other industries, the question of how to achieve a shorter work day within the mines without increasing production costs should receive our serious consideration, the growing difficulty in disposing of the world production of basic commodities—coal, iron, copper, lumber and agricultural products—as well as manufactured goods, now occupying the attention of economists, legislators, labor leaders and society. If the work day is to be shortened, industries such as coal mining, employing labor in its least productive form—lifting with a hand shovel—will, from the fact that individual productivity is held down, fall further behind the more progressive industries. The question we may well ask ourselves is this: Would not a fully mechanized property, employed for three shifts of 6 or 7 hr. each, with resultant reductions in cost per ton, for interest, taxes, depreciation, ventilation, pumping, timbering, etc., make for an even lower cost than would be possible under a hand-loading process, working one shift of 10 hr., as 25 per cent. of our bituminous coal-mine labor was employed a quarter of a century past? This shift, if successful, would transfer a portion of our present mine cost from fixed charges and operating wastes to the pockets of mine labor.

That we are definitely under way is well shown by the record of progress made between 1923 and 1930. It remains for the men who have direct charge of the properties to keep the work going forward, not losing sight of the fact that if it were not for the substitution of other fuels for coal, and the extraordinary economies that have been made in the use of fuel, the coal industry of the nation would be enjoying an annual production of one billion tons today rather than one-half of that amount, which we are now producing. Any attempt to check the onward march of progress represents a mere waste of time, energy and opportunity, and I am glad indeed to know that the anthracite industry, in the renewal of

its wage scale, laid definite stress on the importance of attempting to "eliminate strikes and shut-downs, group action on the part of employees designed to restrict output" and to keep mine labor out of petty politics. The representatives of the anthracite mine workers who have pledged their membership to the "promotion of efficiency" and "production of an improved car of coal" have gone a long way toward the betterment of labor relations. It is safe to say that the new long-term wage agreement made by the anthracite operators and their employees contains the basis of a splendid opportunity not only for the use of more machinery but also for the further extension of human engineering.

DISCUSSION

(*Samuel A. Taylor presiding*)

J. T. Ryan, Pittsburgh, Pa.—Mr. McAuliffe suggests double and triple shifting of fewer mines. I have just returned from visiting the mines in Nova Scotia. Much to my surprise I found that the problem that was troubling the operators in Nova Scotia was an endeavor to get away from double shifting and to go to a single shift. A few days after I returned from Nova Scotia I was in the lower anthracite field, and I found that some strikes there were caused by an attempt to go from single shifting to double shifting. The problem is not easy and some surrounding conditions have a bearing on it.

The reason the operators in Nova Scotia are trying to go from a double shift to a single shift is interesting. The coal is owned entirely by the Province of Nova Scotia, and the company operates under a lease from the provincial government. The manner in which the mines are worked and the working conditions are somewhat under the control of the Department of Mines. The double-shift basis has been established for generations, and the operator cannot shut down a mine. If there is an overproduction, the simplest way out of it normally is to close some mines. That cannot be done in Nova Scotia without the permission of the government. Mines cannot be closed without very good reasons, and so the operators believe that by going from a double shift to a single shift they can keep their mines operating and thus reduce their production.

C. Evans, Jr., Scranton, Pa.—Mr. McAuliffe mentioned the Anthracite Region in the last paragraph of his paper, and it may be interesting, therefore, for me to tell you something of the progress that we are making there in respect to double and triple shifting. Thus far our consistent double shifting has been limited to preparation plants and underground development work in the instance of double shift in pillar removal. The company that I work for has continuously operated two of its large preparation plants on double shift for more than 1½ years. A small preparation plant is operated double shift in order to take the output of a large mine; coal produced in one shift is stored in railroad cars and redumped in the second shift. There is practically the same schedule at another preparation plant, to which coal from two mines is being shipped.

Another large company, the Philadelphia Reading Co., is doing the same thing on a large scale at its newest breaker, and we understand that it is planning to extend such operations.

I think what Mr. McAuliffe had in mind was the double-shift operation of the mine itself. It is interesting to note that in our regions, as our mines approach exhaustion, we have been forced by economic pressure to double and triple shifting.

At first we thought it was going to be uneconomical and we resisted the pressure for a number of years, but finally we had to face the situation. Now, in all of our smaller mines that are approaching exhaustion we are working at least a double shift and in one a triple shift underground. The mine that is working a triple shift is doing it in order to minimize the damage done by surface subsidence. At that mine we are dropping pillars underneath the populated section of the town, and find that we do very much less damage to the town by triple-shifting the pillars. The pillar removal is now being done in three 8-hr. shifts working continuously except on Sunday, and the coal is being pulled out from under houses so quickly that the occupants hardly realize it.

H. E. NOLD, Columbus, Ohio.—What would be the effect on the ventilation problem of double and triple shifting in some of our very gaseous mines?

E. McAULIFFE.—I believe it would improve the situation. I think the records show that a great many explosions have occurred just after starting time in the morning. If we never had a starting time, but had continuous operation, the multiplicity of things that happen during the off-work period would be taken care of and our safety situation would be improved.

Relative to Mr. Ryan's presentation of the condition in Nova Scotia. Undeniably we will find exceptions to all rules. The matter of government ownership and regulation enters strongly into the operation and control of the mines to which he has referred, which, in addition, very largely have the unenviable peculiarity of being worked under the sea. The combined overburden of rock and water, with perhaps a substantial tidal impact, makes subsidence a very important factor to deal with.

Mr. Evans has presented substantial testimony supporting the theory of double and triple shifting of coal mines. In addition to the advantages he has enjoyed from this method of operation plus the lessened liability of explosion hazard first mentioned by me, may I suggest that the shifting of a substantial portion of the cost now expended in the form of interest, depreciation, taxes and supervision to operation, expressed in certain instances in the form of better wages for the workmen, with the remaining portion of the saving applied to profits, would make for a better and more defensible bituminous coal industry.

Selection of Mechanical Car-loading Equipment

By C. C. Hagenbuch,* Fairmont, W. Va.

(Fairmont Meeting, March, 1931)

Machine loading of coal into mine cars is increasing rapidly. Particular reasons for its use frequently apply to certain localities, but in general, it is profitable to install mechanical coal-loading equipment: (1) when existing hand-loading contract rates prevent further cost reduction; (2) when the height of the coal bed requires that either top or bottom must be taken to place cars in rooms; (3) when labor shortage or limited house-plant capacity prevents the production of desired tonnage; (4) when tonnage requirements are in excess of possible hand-loaded production from developed areas; (5) when working places are scattered and it is possible to concentrate production, and therefore supervision, by the installation of mechanical equipment; (6) when rapid development is essential.

Having decided to install loading equipment, the problem then arises as to what type is best adapted for a particular mine or coal bed, the choice lying between mobile track equipment, pit-car loaders, scrapers, shaker conveyors, belt conveyors and drag conveyors.

Major Factors Influencing Choice of Equipment

In determining the type of equipment to be used, the following major factors must be studied:

Grades.—Adverse grades, while permitting other types of loading equipment to work successfully, may prevent the success of a shaker conveyor installation.

Thickness of Bed.—The thickness of coal governs the choice of equipment. A machine that requires 5 ft. of head room should not be selected for a 3-ft. bed where otherwise it would be unnecessary either to brush top or to lift bottom.

Nature of Pavement.—Soft or scaly bottoms are unsuitable for the use of any type of digging loader, shaker conveyors or scraper loaders. Digging loaders and scraper loaders scale the bottom, mixing it with the coal and causing preparation difficulties; shaker-conveyor drives frequently work themselves loose, destroying the conveyor alignment and creating maintenance problems.

Nature of Roof.—With any type of loading machine, it is policy to obtain the maximum possible tonnage per move per set-up. This must

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be done by providing either a wide or a deep cut, or a combination of both. If the nature of the roof requires the use of crossbars or of an excessive number of posts in order to increase the width of cut, the cost of timber, labor and delays to loading will reduce the saving otherwise made by the installation of mechanical coal-loading equipment.

Mining Systems.—It is extremely improbable that a mining layout which has proved successful for hand-loading will prove adaptable without change for machine loading. After hand-loading has proved what roof, bottom and rib action may be expected under certain conditions, it is advisable to consider a loading machine that possesses adaptability to a system as closely related as possible to the hand-loading system which has proved successful. It is essential that the system selected should provide: (1) safety to the workmen; (2) a maximum amount of coal per set-up; (3) a short distance between loading faces; (4) quick car-changing facilities; (5) near-by side tracks; (6) ability to control the roof and pavement.

Gassy or Nongassy Mines.—All mechanical coal-loading equipment can be used in nongassy mines, but for use in gas symines only equipment which bears the approval of the U. S. Bureau of Mines should be considered.

Fine Dust Stirred into Suspension.—In coal beds where handling creates suspended dust it is advisable, to decrease the explosion hazard, to adopt a type of mechanical loading equipment that will permit the minimum amount of stirring, throwing, dropping, or agitation of coal while it is being loaded.

Impurity Bands.—An impurity band close to the bottom of a bed, if not too hard, may be cut out by a shortwall or longwall mining machine, and the coal loaded out by either mobile or immobile loading machines. If the impurity band is so high in the bed that undercutting machines cannot reach it, and if it is advisable to remove this band by cutting, the employment of track-cutting machines and mobile loaders is advisable.

Size of Product.—Naturally, excessive digging, agitation and dropping cause excessive degradation of coal. Where size is important, careful consideration should be given to the amount of handling necessary with different types of equipment.

Impurity Extraction at Face.—When the coal bed contains free impurities which cannot be removed by the mining machines, it is important that the men working at the face have an opportunity to hand-pick the impurities. Little opportunity is presented for this important function by the class of machines which, we might say, "shovel their own coal." Where hand-loaders shovel the coal on to the loading machine, face preparation equivalent to the ordinary hand-loading preparation may be obtained.

Structure of Coal.—When coal is extremely hard or lumpy, and is mined in large blocks, selection of the proper loading or conveying equip-

ment is difficult. As a rule, such coals have enhanced market value on account of their size. It is important that equipment be selected that will handle these large blocks with minimum degradation.

Size of Mine Car.—When conveyor or scraper loaders are to be used, the size of the mine car is not of paramount importance, as cars are spotted at the loading point in trips and car changes are made without loading interruption. However, when pit-car loaders or track-loading equipment is being considered, the matter of car changing has an important effect on the capacity of the loading machine. Car changing represents one of the largest lost-time items in the operation of these loading machines, and it is evident that a 3-ton car, as compared with a 1½-ton car, will reduce the car-changing losses by 50 per cent.

Cutting Machines Available.—Usually, mechanical loading equipment is installed without the purchase of a new kind of cutting machine. When bottom-cutting machines, which can be operated off the track, are available, no restriction will be placed upon the type of loading machine selected. When track-cutting machines only are available, conveyors or scraper loaders should not be selected.

Maintenance Cost.—Probably maintenance cost is the most important item that affects the saving possible by the installation of mechanical coal-loading equipment. Whenever possible, data should be collected relative to upkeep cost of various types of equipment working under conditions similar to those prevailing at the mine where an installation is proposed.

Effect of Breakdowns on Output.—Naturally, a high-tonnage machine with several motors and intricate mechanism means a greater breakdown hazard than a machine with one motor and a simple mechanical layout. Suppose that we consider the purchase of a large machine that will load 300 tons per shift, and that for the same sum we can buy three machines, each capable of producing 100 tons per shift, and also that for the same expenditure, we can purchase 10 machines, each of which will produce 40 tons per shift. In case of a breakdown with the 300-ton machine, we lose the total output, whereas a breakdown of one of the machines of smaller capacity causes the loss of only a portion of the output.

Rate of Advancement.—When mechanical loading is adopted for developmental purposes, it is important to select a type of machine that will give the maximum advancement in narrow work. The advancement possible is affected by height of coal bed, distance between working faces, nature of roof and so forth.

Tonnage Increase per Man.—It is almost invariably true that mechanical coal-loading installations are considered with the primary idea of making an increased output per man. Therefore the possible percentage increase of machine-loaded tonnage over hand-loaded for various types of machines installable should be given careful consideration.

Possible Reduction of Loading Rates.—An increased output per man will permit scale reductions. A careful investigation and study of possible reductions should be made, together with selection of mode of payment as between day rate, straight contract, or contract and bonus.

Organization.—Mechanical loading equipment will not produce desired results simply by installation and the selection of a mining system. Supervision must be detailed, as companies can no longer countenance superficial observation of processes by foremen. Complete success can be assured only by the whole-hearted cooperation, constant supervision and keen initiative of the entire organization.

Cost Credits and Debits.—Before selecting the type of mechanical loading machine, a clear idea should be obtained of just where cost savings may be expected and of where additional expense may be incurred. The savings, as well as the added expenditures, are not always tangibel. The principal savings are:

1. Decreased loading wage due to increased tonnage per man.
2. Decreased gathering cost where mine cars are spotted in trips instead of individually.
3. Saving in yardage in beds where low height prevents the placement of mine cars in the rooms without brushing.
4. Saving in material and labor due to elimination of switches and track where scrapers or conveyors are used.
5. Saving in gathering haulage, main-line haulage and rock disposal when brushing of top or bottom is eliminated.
6. Saving in track, wire, drainage and ventilation due to decreased territory necessary to be opened to get the same tonnage.
7. Timber and tie savings. Because of rapid extraction of coal and completion of panels, it is seldom necessary to replace ties or timbers on account of deterioration; the original installation serves throughout the life of the panel.
8. Saving in supervision and maintenance due to concentration of tonnage.
9. Capital expenditure savings for house plant due to increased tonnage per worker.
10. Increased sales in times of good market, due to quick tonnage increase in shorter period than possible with hand-loading.

Against the savings just enumerated, the following debit charges for mechanical loading equipment must be considered:

1. Increased power costs due to added demand and energy used.
2. Increased depreciation.
3. Added loading-machine maintenance cost.
4. Cost of moving and setting up conveyors and scraper loaders.
5. When hand-cleaning cannot be accomplished at the face, increased cost of tipple-picking.

6. Under certain conditions, timber requirements in excess of hand-loading.

There have been instances of the abandonment of mechanical installations. Sometimes this has been the result of improper machine selection; for example, shaker-conveyor installations where coal must be conveyed up excessive grades, and machine installations with misfit mining systems, or installation of machines insufficiently strong to withstand the handling of coal of hard structure. However, the fact that mechanization is meeting the coal industry's need for decreased costs is proved by the rapid growth of mechanization installations.

Of the bituminous mines of the United States as a whole, 230 were wholly or partly mechanized for loading in 1929 and in 1930 this number had increased to 310 mines, or 35 per cent. The number of mechanized loading units had increased by 41 per cent. The number of pit-car loaders had increased from 1953 to 2407, or 23 per cent., and the number of conveyors from 253 units to 395 units, or 56 per cent.

In the northern Appalachian area, 22 mines were mechanized in 1929, whereas 35 mines had been either partly or wholly mechanized in 1930. This increase amounts to 60 per cent., while the number of units employed increased 100 per cent.

DISCUSSION

(*Thomas G. Fear presiding*)

T. W. Guy, Charleston, W. Va.—What percentage of coal was mechanically loaded last year?

R. F. Roth, Pittsburgh, Pa.—Five per cent.

H. E. Nold, Columbus, Ohio.—Mr. Hagenbuch has stressed the increased tonnage produced per man due to mechanical loading. I know of experiments with pit-car loaders where materially increased loading capacity per man was obtained but an analysis of the situation afterwards showed that the major portion of the increased loading was not due to the installation of the loaders but to the fact that mine foremen became alert and furnished better car supply at the loading faces than had previously been the practice.

C. C. Hagenbuch.—You are right; at the same time we have men on hand-loading and pit cars in the same sections, all have equal car service and yet the yield from the pit-car loaders is far greater than that of the hand-loaders.

T. G. Fear, Fairmont, W. Va.—Our company is mining 8 per cent. and expects to run about 10 per cent. this year.

H. Emerson, New York, N. Y.—What percentage of the cost is due to mechanical loading and unloading?

T. G. Fear.—In some of our lower cost mining, that part of the cost would be 35 to 40 per cent. of the total labor supply.

H. N. Eavenson, Pittsburgh, Pa.—Usually, if the loading rate is 40¢ per ton the labor cost would be 90¢, to which must be added power, supplies, etc.; an average of 35 to 40 per cent. is close to being correct.

R. F. Roth.—One statement in Mr. Hagenbuch's paper is not clear to me, and I cannot quite agree with him. He says: "For the same expenditure we can purchase 10 machines, each of which will produce 40 tons per shift. In case of a breakdown with the 300-ton machine we lose the total output, whereas a breakdown with one of the machines of smaller capacity causes the loss of only a portion of the output."

As I see his comparison, it would require more territory and more upkeep men with the 10 units than it would with the one large unit. I wonder whether it would not tend to get away from concentration and also get away from lower cost to do that. It might be cheaper to have an extra one of the large units on hand to rush in in case of a breakdown, or at least to keep on hand a supply of the parts most liable to a breakdown or failure from power trouble.

C. C. Hagenbuch.—I am talking about pit-car loaders that may be purchased from $650 to $1300. When I said 10 machines, I put them in at $1000. When I mentioned a 40-ton machine I had in mind a pit-car loader; when I mentioned a 300 to 400-ton machine, I had in mind a mobile type of loading machine, costing in the neighborhood of $10,000. There would be no loss of concentration, as daily tonnage and therefore number of working places is practically the same for either type.

R. F. Roth.—Your comparison would necessitate different mining systems.

C. C. Hagenbuch.—Yes.

T. G. Fear.—Just as much concentration with pit-car loaders as loading machines.

R. M. Lambie, Charleston, W. Va.—Do you make any changes in shooting methods?

C. C. Hagenbuch.—On a paper as general as this I could not go into the blasting system. For pit-car loaders we use the same blasting system as for the hand-loaders. The rooms are the same.

R. M. Lambie.—Mr. Fear, do you know of any records comparable with that of the United States Coal & Coke Co.?

T. G. Fear.—I have had Jeffrey shortwall loaders that have driven 50 ft. in one day. The United States Coal & Coke Co. had six loaders in one place, worked two about two hours and let the others rest. Those records are not worth anything to me until I see the cost. I think what you refer to would cost about three times as much as if mined at ordinary rate.

A. R. Matthews, Fairmont, W. Va.—There is one feature that is always connected with a mechanical loading machine of any type in a coal mine. Invariably the superintendent and mine organization are made to realize that this machine is expected to be a success and that it is up to them to see that it is a success. The first step of the mine organization invariably is to put the best men on the machine, and that machine invariably gets the best attention on supplies and service. It seems to me that we have to show a wider spread in the tonnage per man-day in mechanical loading against hand-loading. Usually the spread is not particularly impressive and often is offset by an increase in indirect labor.

H. E. Nold.—The development and adoption of mechanical loading to reduce costs seems to be a well established present practice in coal mining. The question

is whether anything has been accomplished for the good of the coal industry after the costs have been reduced. In the cases of which I have knowledge, the lessening of production costs has been almost entirely offset by reduction of sales realization. Wherein is the coal industry benefiting in this increased mechanization program?

H. N. Eavenson.—By reducing the cost in mechanical loading, you pass the saving on to your customers. No coal companies are getting anywhere by it; if a saving is made and it has not already been passed on it soon will be.

T. G. Fear.—This mining game at the present time is a game. Apparently we are all on a big ship that is sinking, and those that swim hard and fast may get to shore. I believe that if conditions remain as they have been in the past two or three years no one will reach the shore. Until somebody comes out with a new boat, we are going to have to swim, and swim hard.

Comparison of Accident Hazards in Hand and Mechanical Loading of Coal

By Eugene McAuliffe,* Omaha, Neb.

(New York Meeting, February, 1931)

The mining press, as well as certain federal and state bulletins, refer from time to time to the relative hazards that attach to loading bituminous coal by hand when compared with the so-called "mechanical loading" process. Much of the coal coming under the head of hand-loaded coal is won in part by the use of machinery; the coal-cutting machine is in general use, the electric power drill employed to a lesser extent. Therefore it may be said that the principal difference in the process employed, mechanical loading versus hand loading, lies in the fact that some type of loading machine is employed in the mechanical loading process to lift the coal from the face of the room, place or entry, after it is mined, placing it in the pit cars for transport to the tipple.

It is a common practice to compute accident records on the basis of tons mined per fatal and per nonfatal accident, disregarding the fact that it is possible to produce by the aid of coal-loading machines the same daily, monthly and annual tonnage with a much smaller force than is required where the coal is all shoveled by hand. Inasmuch as the men who are released by the substitution of machinery for hand labor must find employment elsewhere, suffering some measure of hazard in their new occupation, we are of the opinion that the true relative hazard that attaches to the processes of mechanical loading and hand loading can be determined only by using as the basis of comparison "man shifts worked" or "hours of exposure" experienced per accident.

The Union Pacific Coal Co., beginning with the year 1929, undertook to maintain a true record of man shifts worked per compensable accident, giving equal weight to fatal and nonfatal accidents. Accidents lacking sufficient severity to come under the compensable terms of the Workmen's Compensation Act were not included in the compilation, the Act providing that "no compensation, except the expense of medical attention, shall be allowed for the first seven (7) days of disability, unless the incapacity runs beyond the period of twenty-one (21) days, in which case the compensation shall run from the time of the injury."

With the further thought that all accidents suffered by employees engaged in any service, in or about the place where the loading process

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* President, Union Pacific Coal Co. and Washington Union Coal Co.

is carried on, should be charged to the process there used, every accident occurring below ground was carefully investigated and charged either to hand loading or mechanical loading, unless the workman was engaged in a task independent of the loading process. Surface employees were treated as a separate class, their work not allocatable to either loading process.

TABLE 1.—*Classification of Accidents*

Underground Men	Mechanical Loading	Hand Loading	Other	Total	Per Cent.
Drillers	3			3	01.08
Drivers			11	11	03.96
Electricians	1		1	2	00.72
Examiners		1	1	2	00.72
Foremen, asst. mine			2	2	00.72
Laborers, inside			3	3	01.08
Miners and loaders	1	96	7	104	37.41
Machine men	8	1	13	22	07.92
Motormen			3	3	01.08
Machinist	1			1	00.36
Machine bosses	1		2	3	01.08
Mech. loader laborers	54		1	55	19.78
Pump man			1	1	00.36
Prop pullers	1		4	5	01.80
Rope riders	1		18	19	06.83
Shot firers	2		2	4	01.43
Slate picker	1			1	00.36
Timbermen	1		6	7	02.52
Tracklayers			7	7	02.52
Total underground	75	98	82	255	91.73
Surface Men					
Blacksmith			1	1	00.36
Boilermaker			1	1	00.36
Boiler washer			1	1	00.36
Car repairers			3	3	01.08
Carpenter			1	1	00.36
Electrician, outside			1	1	00.36
Dock boss			1	1	00.36
Laborers, outside			4	4	01.43
Tipplemen			6	6	02.16
Truck drivers			3	3	01.08
Welder			1	1	00.36
Total surface			23	23	08.27
Total	75	98	105	278	100.00

Table 1 sets forth the nature of service rendered by employees when injured or killed. It will be observed that 21 accidents suffered by men

not directly engaged in the operation of coal-loading machines were charged to mechanical loading, while two accidents suffered by others than "miners and loaders" were charged to hand loading.

Table 2 sets forth the tonnage loaded, number of compensable accidents suffered, man shifts (8 hr.) worked and man shifts per compensable accident. An accident occurred for each 1190 man shifts of hand loading to 1721 man shifts of mechanical loading. The figures indicate that mechanical loading is 44.6 per cent. safer, computed on an exposure basis, than hand loading.

TABLE 2.—*Accidents in Loading and in Other Work*

	Hand Loading	Mechanical Loading	All Loading
Men Allocated to Loading Process			
Tons loaded	1,249,958	1,810,674	3,060,632
Compensable accidents suffered	98	75	173
Tons loaded per compensable accident	12,755	24,142	17,691
Man shifts worked (8 hr.)	116,685	129,115	245,800
Man shifts worked per compensable accident	1,190	1,721	1,421
Percentage of tonnage loaded	40.8	59.2	100.0
Percentage of compensable accidents	56.6	43.4	100.0
Percentage of man shifts worked	47.5	52.5	100.0
Men Not Allocatable to Loading Process			
Man shifts other workmen employed underground			163,583
Man shifts all surface employees			105,477
Compensable accidents other workmen employed underground			82
Compensable accidents all surface employees			23
Man shifts per compensable accident other workmen underground			1,995
Man shifts per compensable accident all surface employees			4,586
All Men Employed			
Percentage compensable accidents	35.2	27.0	62.2
Percentage compensable accidents occasioned by other underground			29.5
Percentage compensable accidents occasioned by surface work			8.3
Fatal accidents included above	4	5	9
Fatal accidents occasioned by other than loading underground			3
Total fatal accidents			12

A study of tons loaded per compensable accident loaded, using the distribution shown in Table 1, develops ("other" underground and surface accidents excluded) a margin in favor of mechanical loading of 89.2 per

cent., this result including not only the gain in safety derived from the reduction in the number of men employed, but in addition thereto the advantage of increased unit production per man shift worked.

While fatal accidents, when taken alone for a period of one year, in no sense represent a dependable basis of comparison, the number of such accidents, shown by occupations and separated on the basis used in compiling Tables 1 and 2, are shown in Table 3. Only two out of four fatalities chargeable to mechanical loading occurred to men actually engaged in the operation of loading machines.

TABLE 3.—*Fatal Accidents*

Underground Men	Mechanical Loading	Hand Loading	Other	Total
Miners and loaders		5		5
Mech. loader laborers	2			2
Machine man	1			1
Rope rider			1	1
Shot-firers	1		1	2
Tracklayer			1	1
Total	4	5	3	12

When a comparison is made between the man shifts worked per fatal accident allocatable to mechanical and hand loading, we find (disregarding the three deaths occasioned by other occupations) that mechanical loading shows a margin of safety over hand loading of 38.3 per cent., a figure closely comparable to the relative hazards shown for all compensable accidents.

Committed as the writer is to the belief that not less than five years' results are necessary to a dependable comparison, it will be our purpose to carry on the record inaugurated in 1929, until such time as the percentage of coal loaded mechanically so far exceeds that loaded by hand as to make the discontinuation of the record desirable. We do believe, however, that the results shown for the 12 mines operated in 1929, with pitches ranging from 4° to 17°, good roof and bad roof, and with a variety of coal-loading equipment, including shaker conveyors equipped with "Duck-bills," Joy loading machines, large-capacity scrapers and pit-car loaders, are sufficiently representative to indicate that mechanical loading, when properly managed, does afford an increased measure of mine safety to underground workers.

DISCUSSION

(*Howard N. Eavenson presiding*)

C. E. BOCKUS, Dante, Va., asked to what extent Mr. McAuliffe attributes the increased safety obtained under mechanical loading to better supervision of the men while at work. Mr. McAuliffe replied that undoubtedly increased supervision is

the greatest contributing factor, and although mine discipline is still far from perfect, 90 per cent. of the improvement is due to the closer supervision made possible by mechanical loading because the men work in less scattered areas.

C. Evans, Jr., Scranton, Pa., inquired about the accidents to machine men noted in Table 1, pointing out that eight machine men were injured under mechanical loading while but one was injured in hand loading. Mr. McAuliffe replied that while the figures appear inconsistent when compared with the end result of the study, the particular situation referred to was largely a matter of accident, although it was more difficult to move mining machines in the Rock Springs district under mechanical loading conditions because no room tracks are employed where mechanical loading prevails. Mr. McAuliffe made the further statement that each accident report was scrutinized carefully and the best possible judgment was employed in determining the branch of service against which each accident should be charged.

E. W. Parker, Philadelphia, Pa., inquired whether the loading-machine operators were of a higher grade than the men employed in the hand loading. Mr. McAuliffe replied that while workers in mechanical loading may not be of higher grade, usually they are somewhat younger and perhaps more alert. The very old employees have been provided with pit-car loaders, reducing mechanical skill to the minimum. Mr. Evans asked whether a record of the days lost from accidents is being kept. Mr. McAuliffe replied that accident severity figures are being maintained, but that the results were not incorporated in the paper.

Stripping in the Anthracite Region

By H. H. Otto, Scranton, Pa.*

(Wilkes-Barre Meeting, May, 1931)

Fourteen years ago, J. B. Warriner presented before the Institute a paper on anthracite stripping,[1] describing the progress of stripping in the Anthracite Region from its beginning with an old quarry stripping at Summit Hill, which was operated more than a century ago, through various types of strippings used in 1917. This paper will briefly review the status of anthracite stripping methods from 1917 to the present day and describe in detail the stripping operation of The Hudson Coal Co. at Clinton colliery. It is not intended as a review of the work in the district—that would require a much longer paper.

History and Statistics

Mr. Warriner said, in part: "The sizes of shovels have gradually increased until now 70 and 80-ton shovels are the rule and larger equipment is coming in." He also referred to a large electrically operated dragline which weighed about 225 tons and had an operating radius of 125 ft. In 1917, a number of strippings used shovels of various sizes, ranging up to the 60 Marion and Bucyrus 70-C railroad type. Material was hauled away in narrow-gage side dumpers, usually of 4 or 5 cu. yd. capacities. Star-type churn drills with 4-in. bits were generally used for drilling purposes.

Since 1917 wages in the anthracite fields have increased, but the higher operating costs of strippings are now being successfully combated by continuous improvements in stripping equipment, such as the development of the caterpillar or crawler shovels, the use of larger cars, better drilling equipment, the introduction of shovels of increased capacity and the adoption, in a number of instances, of standard-gage equipment for transportation.

In 1918 another large dragline was introduced in the southern fields, which had been in service in the Middle West and is in use today, dipping vertically about 180 ft. to raise coal out of a stripping. About 1918, small shovels with caterpillar traction were introduced. In 1922 the 6-in. well drills began replacing the 4-in. drills, with an attendant saving to the operators. About the same time A. E. Dick placed the first pair

* Mining engineer, The Hudson Coal Co.

[1] J. B. Warriner: Anthracite Stripping. *Trans.* A. I. M. E. (1917) **57**, 159.

of caterpillars under a railroad-type shovel in the Anthracite Region at Audenried. This was a decided improvement over the railroad type but it was a makeshift arrangement. A number of others were converted in a similar manner throughout the region before the manufacturers were able to develop a 2 to 3-yd. full revolving shovel with caterpillar or crawler traction and market it at a reasonable cost. This made obsolete the railroad-type shovel, so popular in 1917.

In 1924 another big step was taken when the Lehigh Coal & Navigation Co. decided to buy a 320 Bucyrus electric shovel of 6 cu. yd. capacity, 6-in. well drills and standard-gage equipment, including air-dump cars, for its Crystal Ridge stripping near Hazleton. This rangy shovel has no companion in the field at present because early in 1926 the speedy 4-cu. yd. full revolving shovel for quarry purposes was developed. This type of shovel, which costs approximately one-half the price of the 6 to 8-cu. yd. type, became more popular. The 4-cu. yd. shovel has the distinct disadvantages of requiring more track shifting and taking a smaller cut. However, in many cases this is not objectionable because often the size of a job does not warrant the higher capital investment of the larger equipment. The steam shovels throughout the country have gradually given way to the electric, gas and oil-powered shovels. Today we find that this is an aid and makes possible the stripping of isolated coal areas where water is scarce.

In transportation problems we find no rule to govern strippings as a whole, but we are rather amazed at the variety of transportation equipment, such as autotrucks, trucks pulled by caterpillar tractors, narrow-gage cars (hand and air-operated) and the 30-cu. yd. air-dump standard-gage cars propelled by rod or geared locomotives. In addition to steam locomotives, gasoline and electric locomotives are being used to a limited extent in connection with strippings. We now find the trend toward the complete elimination of the use of locomotives and cars for the removal of overburden, and this brings us to the particular job to be described here.

Operation at Clinton Colliery

At Clinton colliery of The Hudson Coal Co., at the extreme upper end of the Northern Basin of the Anthracite Region, the beds are comparatively flat. Parts of the original beds were washed away by glacial erosion, while other areas were left with little or no rock cover, which made it impracticable to mine by ordinary underground methods. It is probable that numerous areas of this type exist in the Lackawanna and Wyoming regions. There are several on Clinton property. Some of the areas have been known for a number of years, others are still unproved.

Stripping prices during the World War and the post-war periods were too high to enable an economical recovery of the coal we are now stripping.

However, the remarkable improvements in excavating and transportation equipment, together with better drilling units and improved methods employed in the use of explosives, have reduced unit costs until prices today are the lowest they have been in 20 years. The most revolutionary step in connection with stripping equipment has been the development of the dragline excavator, originally mounted upon loose wooden rollers then on railroad trucks and later on equipped with caterpillars or else a walking device. This type of excavator is a product of the Monighan Manufacturing Co. of Chicago. It has been used along the Mississippi River and elsewhere for soft digging but is new in the Anthracite Region of Pennsylvania, where heavy overburden and hard shales and rocks are encountered.

Description of Areas and Methods of Operation

Fig. 1 shows the stripping areas with reference to each other and to the tipple and railroad tracks at Clinton. The main Grassy bed stripping area is divided into two parts by the Richmondale branch of the N. Y. O. & W. R. R. The distance from the edge of the stripping to the tipple is 5500 ft. and the extreme length of the whole area to be stripped is about 3600 ft., including doubtful areas, which will be uncovered if the quality of coal warrants. The areas under contract cover 28.3 acres and contain approximately 174,000 market tons of coal, from which approximately 1,112,000 cu. yd. of overburden will have to be removed. The overburden averages 25 ft. in thickness, the coal seam is 5 ft. 5 in. high and contains 4 ft. 2 in. of coal. The cross-sections in Fig. 2 indicate the character and depth of the overburden and the thickness of the coal.

The successful bidders on the Clinton work were Carey, Baxter and Kennedy of New York City, who started to excavate on Jan. 16, 1931. The overburden is being removed by a Monighan dragline excavator equipped with a 160-ft. boom, and a 6-cu. yd. bucket. When the work was begun, this was the largest walking machine that had yet been built. Among the distinctive features in connection with this massive machine is its walking device, which is a decided change from the caterpillar or crawler type of locomotion.

Fig. 3 shows the plan of operating the stripping. The overburden in the area marked *A* was removed from the stripping area by the long-reaching Monighan excavator, which deposits the spoil about 70 ft. from the stripping edge in a pile 55 ft. high. The rock was then drilled, blasted and removed in the second operation by the same machine, thus leaving the coal exposed. The coal was removed and loaded into mine cars by a Bucyrus-Erie E-2 shovel and the area was ready to receive the overburden from the second zone *B*. In a similar manner, the whole area will be uncovered and the coal removed. The excavation in area *A* is 130 ft. wide atthe top and about 75 ft. at the bottom.

Two 20-ton gasoline locomotives deliver the loaded mine cars to a turnout outside the stripping limits, from which the colliery locomotive hauls them in trips to the tipple. The location of the turnout is shown on Fig. 1.

FIG. 1.—LOCATION OF STRIPPING AREAS WITH RESPECT TO TIPPLE, CLINTON COLLIERY, GRASSY BED.

The rangy dragline deposits all of the overburden at its final destination, eliminating the necessity of transporting any of it in cars or of handling it by means of any other equipment.

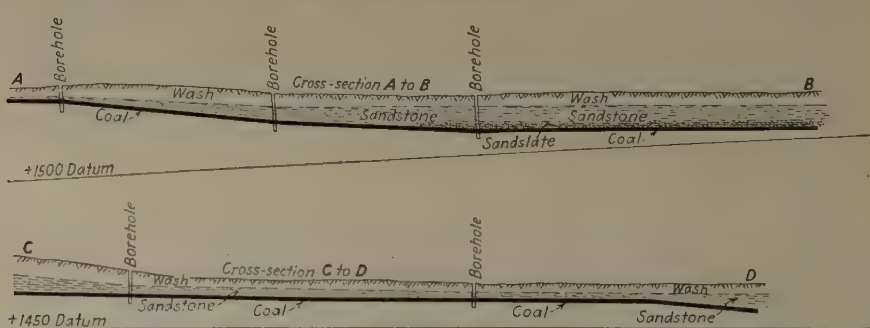

FIG. 2.—CROSS-SECTIONS THROUGH STRIPPING AREAS, CLINTON COLLIERY, GRASSY BED.

An area of Top Clark bed, without rock cover, near the Clinton colliery office, containing approximately 41,000 market tons of coal, is also included in the contract. This area, which is seperated from the main

stripping by a distance of 6000 ft., will be operated in conjunction with the larger area. The Monighan dragline excavator will "walk" to the Top Clark location, strip the area, and "walk" back to complete the main job. It is estimated that the machine will traverse the distance (one way) in 14 hr. It would require at least 30 hr. to move a shovel of the railroad type over the same route, and the labor cost would be very high.

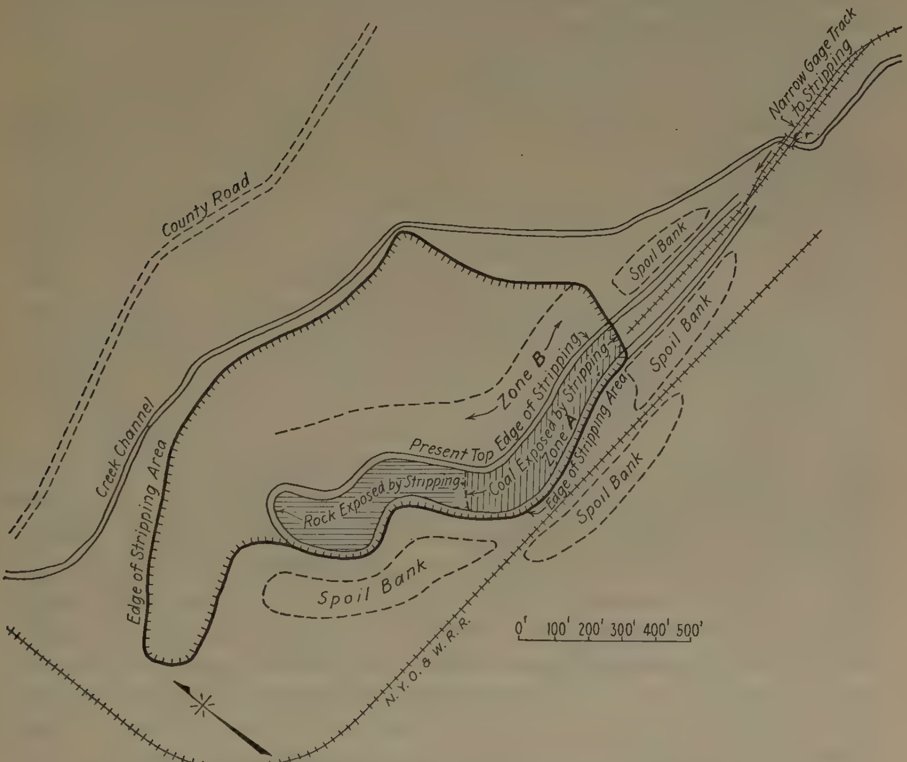

FIG. 3.—PLAN OF STRIPPING, CLINTON COLLIERY, GRASSY BED.

Equipment

1 Monighan dragline excavator, model 6150, 375-ton, equipped with 160-ft. boom and 6-cu. yd. Page bucket. Included in its equipment are one low-pressure and one high-pressure compressor. This excavator is operated by a 300-hp. Fairbanks Morse Deisel engine, which drives the hoist, drag and walking device.

1 Bucyrus-Erie Deisel-driven shovel, model E-2, 50-ton, convertible to dragline or crane. The shovel bucket capacity is 1¼ cu. yd., while the dragline bucket has a capacity of 1 cu. yd. The shovel boom is 29 ft. 6 in. long and the dragline boom is 50 ft.

2 Whitcomb gasoline locomotives, 20-ton.
1 Model 50 Monarch tractor equipped with bulldozer.
1 Athey trusswheel wagon, 10-ton.
4 type X-59 Ingersoll-Rand jackhammer drills.
1 type R-39 Ingersoll-Rand jackhammer drill.
2 railroad tank cars, 12,000-gal.
2 Deisel-driven direct-connected air compressors installed on railroad cars
2 standard-gage supply cars.
1 flatcar for miscellaneous supplies.
1 office car.
1 blacksmith shop.
1 Ford truck.
3 all-metal garages.
1 power house.
Oil line, 1½ to 2-in. dia., extending from tank car to filling station.
Air lines; 4-in. gas pipe originating at compressor plant and extending along the edge of the stripping with 2-in. branches extended from the main line to the drills wherever needed. Flexible dresser couplings, which are easy to install, are being used.

The Excavator

The Monighan excavator has two feet, or "walking platform shoes," each of which covers an area of 210 sq. ft. When the huge machine is operating, it rests on a circular bearing base of 30 ft. 8 in. dia. and the shoes are suspended about 2 ft. above the ground. The bearing weight on the walking platforms is 6½ lb. per sq. in. The walking shaft extends across the revolving platform, is fitted with a large upset square shoulder for the walking gear, and at each end terminates in a cam. The walking gear receives its power through a heavy forged cut tooth clutch pinion on the end of the loading drum shaft. This pinion is provided with a jaw clutch and hand brake. When ready to move, the clutch is thrown in by the operator and power applied to the main driving shaft, which lowers the shoes to the ground. As soon as the shoes strike the ground, the circular base is raised and moved ahead 7 ft., and lowered gently to the ground by means of the cams.

Advantages of the walking device are that the excavator can be moved forward, backward or in any direction; it can turn quickly, and almost as abruptly as a man pivoting on his heel.

The swing is electrically driven and when handling surface material 1 min., 10 sec. to 1 min. 30 sec. is required to complete a cycle, including a swing through an arc of 180°. On hardpan and rock, the speed is reduced to approximately 1 min., 30 sec. to 1 min., 50 sec. per cycle, the variation in time being governed by the nature and size of the pieces being moved. The recording charts, which are kept on file, show that, with

delays included, the time required for the shovel to complete a cycle averages approximately 1½ min., of which about 40 sec. are consumed in loading and in the neighborhood of 50 sec. in swinging and dumping.

Classification of Material

The definitions of earth and rock vary. Most variations in classifications occur at the gradation zone between earth and rock, where the material to be excavated is soft and of the consistency of earth.

The contract under which this stripping is being done specifies a division of material into two parts—common excavation and rock excavation. Common excavation is defined as earth, clay, sand, gravel and boulders, as well as stratified material which can be handled without blasting. Rock excavation is defined as stratified rock in solid beds in its original position, which cannot be removed without blasting.

Drainage

The Bucyrus-Erie dragline was used to construct a ditch around the stripping area to take care of the hillside water and also for the diversion of a small stream which has been destroyed by the stripping operation.

All water within the stripped area of the Grassy bed drains through the broken strata to the Bottom Clark bed, which is the lowest seam to have been robbed beneath this area. There it finds its way to a permanent water ditch through which it flows to the surface at Wilson Creek, a point about 2 miles south of the stripping area.

Drilling and Blasting

Drill holes for blasting are arranged in parallel rows with the holes in each row staggered on 8-ft. centers and with the depths ranging from 5 to 18 ft., depending on the thickness of rock. The compressed air for drilling is furnished by two air compressors installed in a box car, driven by direct-connected Deisel engines and operating under 90-lb. pressure. Four type X-59 Ingersoll-Rand jackhammer drills are used in drilling the rock, and one type R-39 is used on the coal. For drilling, 1¼-in. round steel is used and the starters range from 2¾ to 1⅝ in. dia. For blasting the rock, 40 per cent. quarry gelatin, 1½ by 8 in., is used exclusively, and an average of ½ lb. of dynamite is used per cubic yard of rock excavated. One No. 6 electric exploder is inserted into the top stick of dynamite and the remaining opening is tamped with the cuttings from the hole.

A 100-hole blasting battery is used in firing, 100 holes being connected in series and fired at one time. The rock is blasted to such fineness that there is no trouble in dragging it off the coal. Black powder and pellet powder are used for shooting the coal.

Maximum Operation Force

Supervision

1 superintendent
1 engineer and office man
1 master mechanic

Monighan Dragline Excavator Crew

The dragline operates two shifts with one operator and one oiler on each shift.

2 operators
2 oilers

Bucyrus-Erie Shovel Crew

1 operator
1 oiler

Drilling Crew

1 drill boss
4 drillers
4 helpers
1 nipper

Miscellaneous Labor

1 blacksmith
1 blacksmith's helper
1 compressor operator
1 truck driver
2 tracklayers
2 car men for cleaning refuse

In 1917 Mr. Warriner said that the average force required to operate a one-shovel stripping consisted of about 35 men. Today, on the Clinton job an average of 18 men are being employed, while the pay roll includes a maximum of about 27 men.

Mr. Warriner also made the following statement concerning the output of shovels in 1917: "Under proper conditions outputs as high as 30,000 cu. yd. per month have been obtained for one shovel in clay. The average, however, is only about 18,000 cu. yd. for clay and 10,000 to 12,000 for rock." At the Clinton stripping, during the month of March, the excavator moved 78,000 cu. yd. of material, of which 51,500 cu. yd. were clay and 26,500 cu. yd. were rock; and during April, 85,700 cu. yd. of overburden and 1,400 market tons of coal were removed. The dragline excavator was worked on a double-shift basis in order to obtain this yardage and the delays averaged approximately 5 per cent. of the working time.

Loree Red Ash Stripping

In striking contrast to the operation at Clinton is a small stripping on the Red Ash bed outcrop at Loree colliery of The Hudson Coal Co. at Plymouth. Here the roof rock is too weak to afford proper protection for ordinary underground mining and the coal was left unmined when the pillar robbing was started. A geological disturbance in the form of an upthrow displaced the Red Ash bed near the crop so that it is almost vertical; the average pitch in this area is 50°, compared to the normal pitch of the bed, which is between 7 and 12°. Because of this upthrow, the stripping had to be developed in two sections.

Robbing having cut off access to underground transportation routes, bids for the stripping were requested on the basis that the coal was to

be delivered by means of motor trucks, to the breaker tipple, a distance of 1½ mile from the area to be stripped. The narrow-gage tracks on the surface could have been extended to the stripping but trucking was chosen because of its mobility. An advantage of this company is that it has the unhampered use of mine tracks and their facilities, and the contractor alone is responsible for his success or failure in getting the coal to the tipple.

The stripping is being done by Dewees Brothers of Philadelphia. It was started on Aug. 12, 1930. The stripping area originally contained about 35,500 marketable tons of coal, the vein averaging 12 ft. in thickness. The overburden approximated 259,900 cu. yd. Three 1¼-cu. yd. shovels load the overburden and coal into a fleet of 14 trucks, which have capacities varying from 4½ to 7 cu. yd. The overburden is trucked about ½ mile to an old stripping excavation that is being filled and leveled. There are some heavy grades between the stripping and the spoil bank. The grade at the outlet from the stripping averages about 20 per cent. against the load.

The removal of overburden has proceeded almost without interruption from the commencement of operations, and an average of 24,000 cu. yd. per month has been handled. Up to May 1, 1931, a total of 209,900 cu. yd. of overburden and 29,600 market tons of coal had been removed. On account of market conditions, the colliery has been working broken time, but this has enabled it to build up a coal reserve at the stripping to be dumped on colliery working days.

The best output obtained during the period of operation was in the month of February, when an average of 127 truck loads, or 601 market tons, of coal per day was produced. Considering the pitch of the coal and the wet and snowy weather, this was a good performance and helped the colliery output when the market demand was high. This was accomplished without hindrance to our normal colliery operation.

We believe that this type of equipment will solve the problems of operation in the smaller isolated stripping areas.

Acknowledgments

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Premature and Hangfire Explosions in Anthracite Mines

By Charles W. Wagner,* Scranton, Pa.

(New York Meeting, February, 1931)

A premature explosion might be described as an explosion that occurs before the miner expects it. Notwithstanding that it is unexpected, a premature is generally within the miner's control. A hangfire explosion is a blast that occurs after the miner expects it, and in all instances, except where the explosives or blasting accessories are defective, hangfire explosions are within the miner's control.

Causes of Premature Explosions

Generally, there are seven methods of shot-firing in anthracite mines; namely: (1) cap and fuse with dynamite, (2) fuse with pellet powder, (3) squib with black powder or pellets, (4) exploders and delays with dynamite or black powder, (5) exploders with dynamite, (6) delay ignitors with black powder or pellets, (7) electric squibs with black powder or pellets.

Cap and Fuse.—This method of firing seems to produce the greatest number of premature explosions. This is not solely because of the method, but is the result of mispractices which are possible with this method of firing. The two most common mispractices followed with cap-and-fuse blasting are the firing of multiple holes and the use of short fuse. In the firing of multiple holes by cap and fuse, investigations have shown that in nearly every instance where the miner was injured by this method of firing he remained too long at the face lighting the several fuses, so that the first shot went off before he could reach a place of safety. Such accidents could have been avoided either by accurate judgment by the miner of the relative lengths of the fuses or by the use of electric firing. Rules exist in nearly every mine forbidding the lighting of more than one hole at a time where the cap-and-fuse method of firing is practiced. Where it is desirable to fire more than one hole at a time, electric firing should be used.

It is difficult to understand why so many premature explosions result from the use of short fuses, inasmuch as the cost of fuse is practically a negligible item in the expense of a day's work. Notwithstanding this, however, we frequently find miners using short lengths of fuse in an effort to save one or two cents per hole, or about 15 to 20¢ in a day. A practice

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has grown up of putting about 18 in. of fuse on the primer, and inasmuch as this 18-in. piece of fuse could not project out of the mouth of the hole, the miner first lights the fuse and then shoves the primer into the hole. As the burning time of fuse is about 35 sec. to the foot, this method of firing frequently results in a premature explosion, which often injures the miner before he can reach a place of safety. Aside from the danger involved, the miner overlooks the fact that the "economy" he hopes to effect by reducing the length of the fuse is generally offset by the additional amount of explosive necessary, because such a hole must necessarily fire untamped. While considerable effort is being made in the anthracite field to curb this dangerous practice, many miners continue to take such chances when they believe that the boss is not near by.

Fuse with Pellet Powder.—The same mispractices of lighting multiple holes and using short fuse as were described with the use of cap and fuse are permitted by this method of shot-firing.

Squib with Black Powder or Pellets.—The use of squibs with black powder in cartridges, or squibs with pellet powder, permits the mispractice of tampering with the squib. The ordinary miner's squib is very delicate, and unless in the hands of an expert it may become a great source of danger.

Exploders and Delays.—These are used for the purpose of firing more than one hole at a time. The economy of this practice is apparent, especially in mechanical mining work, heavy pitch work and in places where ventilation is poor. The principal cause of accidents through the use of delays is in the use of exploders and delays of different bridge-wire resistances in the same series.

This can be illustrated by citing a specific case: In the particular instance in question, the miner connected instantaneous electric blasting caps of 1.1 ohms resistance in the same series with delays of 0.9 ohm resistance. After twisting the blasting machine and not hearing the blast, the miner, after waiting what he considered a reasonable length of time, returned to the face and was caught by the blast from the No. 6 delay primed charge. Subsequent investigation showed that there were differences in bridge-wire resistances and also a difference in ignition points of the explosive mixtures in the exploders and delays; also, that the lead wires were bare in several places. The blasting machine, a 10-hole Schaeffler, was in first-class condition.

The conclusion reached after this investigation was that the bare wire grounded the circuit to such an extent that the current could affect only the bridge wire of low resistance; this happened to be the No. 6 delay, or the first one in the series, which also contained a detonating agent with the lowest ignition point. Not only did the No. 6 delay in this instance have the lowest resistance; it also had a detonating mixture surrounding the bridge wire which had a lower ignition point than the

mixture of the other delays and exploders in the series. We believe that this accident could have been prevented if the miner had used exploders and delays of the same bridge-wire resistance, or in other words, if he had used exploders and delays manufactured by the same powder company.

Frequently several powder companies supply the explosives needs of the same colliery, and the exploders and delays are likely to be mixed, so that a miner might have exploders and delays manufactured by two or more powder companies. Each powder company recommends that only its own blasting supplies be used in the same circuit or in the same blast. While at first this may appear to be a sales slogan, it possesses sound foundation; the bridge-wire resistance and the ignition points of various exploders and delays are the same for a particular powder company, but vary considerably between the individual manufacturers. Where more than one powder company delivers explosives to a colliery, special care should be exercised to see that miners are supplied with only one make of exploders and delays.

Dynamite Exploders, Delay Ignitors and Electric Squibs.—Using exploders with dynamite, delay ignitors with black powder or pellets and electric squibs with black powder or pellets, there is not much danger from premature explosions except when the miner fails to remove the lead wires from the source of current when charging a drill hole or violates some general rule when charging.

Electric Currents and Detonators

In all methods depending upon the use of electric current, which includes the last four methods of firing, there is a common hazard which is caused by an unexpected current running through the firing circuit before the miner has a chance to reach a place of safety. Such unexpected currents might be caused by someone handling the firing device while the miner is at work at the face, or by stray currents through compressed-air pipes, bottom rock, rails, etc. Accidents from these causes can be avoided or minimized by the use of shunted exploders and delays and by taking proper precautions to use well-insulated lead wires and connecting wires. In addition, binding posts on all blasting machines or batteries should be made so as to prevent the attachment of wires except when the operator is ready to set off the blast.

Firing by means of blasting caps, electric blasting caps and delays has a hazard in the fact that some miners insist upon inserting these detonators diagonally into the primed cartridge. Unless this is done with the greatest care, probably one end of the detonator will extend beyond the circumference of the cartridge, and by friction against the side wall of the drill hole may be set off while the miner is inserting the cartridge or tamping the hole. We believe that the powder companies advocate insertion of the cap or exploder parallel to the axis of the cartridge.

This eliminates the danger just described and also effects greater efficiency in the detonation of the charge in the hole.

Causes of Hangfire Explosions

Regardless of the method of firing, hangfires occur. The three principal causes are as follows: (1) Improper cleaning of the drill hole, so that drillings and fine dust collect between the cartridges of explosive when charging; (2) weak detonators; (3) insensitive explosives.

Improperly Cleaned Drill Hole.—Drillings or dust cause a separation between the individual cartridges of explosives, which acts as an insulator between the priming cartridge and the remaining cartridges, so that while the primer may explode, it will not transmit the explosion to the succeeding cartridge or cartridges because of the cushioning and insulating effect of the dirt between the cartridges of explosives. While the remaining cartridges may not be immediately detonated, it is probable that sufficient heat will have been generated by the explosion of the primer to cause the cartridge next to it to burn. The heat set up through the burning of the unexploded dynamite will build up sufficient heat and pressure finally to explode the remaining cartridges in the hole. A miner, hearing the priming cartridge explode, might believe that the hole was completely exploded and, after waiting a reasonable length of time, return to the hole only to be caught by the subsequent explosion of the remaining cartridges. Such an accident could be avoided, or at least minimized, by proper care in cleaning out the hole. Sometimes the work of a scraper or spoon must be supplemented by the use of compresssed air.

Weak Detonators and Insensitive Explosives.—Occasionally hangfires result from weak detonators and insensitive explosives. A weak detonator fires only a portion of the charge and sets up a condition similar to that already described, wherein the miner failed to properly clean out the drill hole. That is to say, the explosion of a small portion of the total charge sets the remaining portion of the charge on fire, and, because the miner has heard an explosion, he may return in time to be caught by the blast of the portion of the explosive in the drill hole that has been set off after a delay by the burning of a part of the explosive. Insensitive explosives act almost in the same way as dirty drill holes and weak detonators. Generally speaking, only a few of the accidents are due to improper explosives and blasting accessories.

Summary

Practically all of the accidents caused by firing could be avoided if the miner followed safe practices. The first suggestion of the layman for the cure of these evils is to lay down definite and drastic rules accompanied by more or less severe discipline, but the American public has learned within the past few years that people cannot be forced to do

things simply because there is a law on the statute books. Much more can be accomplished by an intensive educational program under the direct supervision of the foreman or section foreman in charge of the particular mine or miners. Discipline should only be resorted to when a miner persists in violating the instructions of his boss, or where discipline will have a good effect upon all of the men at the colliery. The mere making of rules and the application of discipline without education will make it necessary to police every workman in the mines.

It is hoped that this paper will be the nucleus of a wide discussion and that sufficient constructive criticism will be offered to enable anthracite companies and powder manufacturers to formulate standard practices, which will result in the elimination of accidents resulting from the mishandling of explosives.

DISCUSSION

(*Howard N. Eavenson presiding*)

S. P. Howell, Washington, D. C., discussed the U. S. Bureau of Mines method of classifying and tabulating accidents resulting from explosives, and cited statistics to show that premature shots are the most frequent cause of such accidents. He also said that Bureau statistics show that electric firing of shots is far the best from a safety point of view, because it gets the shot-firer safely away from the flying particles. However, the current used for firing has sometimes been known to cause gas explosions, the use of the multiple-shot battery being unfavorable in this respect. These explosions may be due to bare lead wires, short circuits or from the ends of the lead wires shorting to the ground instantly after the blast.

S. S. Ellsworth, Simsbury, Conn., asked if the dangerous practice of using too short a fuse could be overcome as easily as to substitute electric blasting for fuses. Mr. Wagner said that it is very difficult to get the miner to use a sufficiently long fuse, as he saves 1 or 2¢ per hole by cutting them as short as possible. It is difficult to promote electric firing because of its greater expense, but the Glen Alden Coal Co. has one mine in which 27 per cent. of the firing is done electrically.

C. Evans, Jr., Scranton, Pa., asked whether in a mine using explosives from two different manufacturers it is better to use accessories from only one or from both manufacturers. Mr. Wagner favors a single brand of accessories but if two are necessary they should not be mixed.

C. S. Hurter, Wilmington, Del., said that elsewhere, particularly in Great Britain, the greater number of accidents occur in loading and tamping the hole. Mr. Wagner replied that here most of them occur in firing the shots.

J. J. Rutledge, Baltimore, Md., expressed the opinion that the introduction of the fuse marks the real beginning of blasting troubles, as it is so subject to abuse, particularly by short fusing. However, some of the more progressive mining companies are supplying their men with fuses in standard lengths and the mine inspector sends men out who are found with shorter lengths in their possession. He recommended the manufacture of a fuse of distinctive color, not easily discolored, like the old Cornish fuse, so that the miner would not easily make a mistake in cutting his fuse. He believes that the miner who gives the foreman trouble by using short fuse

will also cause trouble and be careless in using electrical shot-firing, because the unsatisfactory results in both cases are due to inherent carelessness on the part of the miner.

C. R. Seymour, Simsbury, Conn., regarded the moment of greatest danger as that in which the fuse spits and showers out sparks just as it is lighted. These sparks are apt to fall on the explosive itself and cause a premature explosion. Mr. Wagner expressed doubt as to the magnitude of this risk.

S. P. Howell added that while the short fuse is the greatest single danger, perhaps, it is not the only one, men often being injured by trying to relight a fuse which they thought had not lighted in their first attempt, but which actually had done so. All methods of firing must be regarded as dangerous, but accidents may be largely eliminated by inspection and education.

H. G. Turner, Bethlehem, Pa., suggested that uncuttable fuses might be made which could be furnished to the miner in proper lengths only, thereby preventing short fusing.

C. S. Hurter directed attention to the relative merits of loose-core and solid-core fuses, saying that the latter sometimes give a false effect of ignition owing to powder sticking to the cotton core thread.

Economic Aspects of Bituminous Coal Losses in Ohio, Pennsylvania and West Virginia

By James D. Sisler,* Morgantown, W. Va.

(Fairmont Meeting, March, 1931)

Among the various studies made by the Coal Fact Finding Commission, appointed by President Warren G. Harding in 1922 to investigate all phases of the coal-mining industry in the United States and to report to Congress, was one concerning the amount and nature of losses in mining bituminous coal in the eastern part of the United States. This study was carried on under the direction of the U. S. Bureau of Mines. The writer was appointed field engineer to investigate the problem in Ohio, Pennsylvania and West Virginia. Some 400 mines were studied in these three states and conclusions were drawn from many sources of data. The summary report written by George S. Rice and J. W. Paul states that "the field reports indicate that the recovery in Ohio will remain about as at present, except in the Belmont district; whereas in Pennsylvania it will gradually improve, and in West Virginia it will improve rapidly." Since 1922, the writer has been interested in gathering material to substantiate or contradict these prophecies.

Nature of Coal Losses

There are six general causes of losses of coal in mining: (1) coal left on the roof and bottom; (2) coal lost in room, entry and panel pillars; (3) coal lost in oil-well or gas-well pillars; (4) coal lost under buildings, railroads and boundaries; (5) coal lost in handling and preparation, underground and surface; (6) coal lost by rolls, thin or dirty areas and under streams.

Local economic aspects that dominate the type of mining conditions determine to some extent whether certain losses are avoidable or unavoidable. In general, however, unavoidable losses include coal lost in the protection of buildings, streams, railroads, restricted properties and around oil and gas wells. Barrier pillars and other safety pillars are unavoidable losses as long as their size does not exceed the minimum for adequate safety. Natural physical conditions beyond human control, such as rock faults, clay veins, partings and binders, bad top and bottom, rolls and horsebacks, cause unavoidable losses. Unavoidable losses

* State Geologist, West Virginia Geological Survey.

were classed by the Commission as "those which cannot be overcome without undue danger to life, without undue labor per ton of product and without serious damage to buildings and surface properties." Avoidable losses are principally those which are directly caused by improper mining methods, mine plans, supervision and design of equipment. The causes of avoidable coal losses are practically all economic and are numerous.

Causes of Coal Losses

Pillars.—In districts where the initial cost of coal is low, pillars are not pulled. Since the slump in the coal business many operators of small mines have completely charged off the initial cost of the acreage and are mining only the thickest and best coal in order to ship a cheap product. The loss from this cause increases in direct ratio to the strength of market competition. Operators of large plants, particularly in Pennsylvania and West Virginia, are recovering more coal from pillars than was recovered in 1922. This increase in recovery offsets the increased loss of coal from this cause in the smaller mines. The pillar recovery in Ohio is practically the same as it was in 1922. On a tonnage basis West Virginia and Pennsylvania have increased the percentage of pillar recovery since that date.

Squeezes.—In former years rooms were driven too wide and pillars were left too narrow; consequently squeezes started. Squeezes are induced by rib gouging in both machine and pick mines; with better engineering standards has come a decrease in loss from this cause. Squeezes are more frequent when the market is booming and strenuous efforts are being made to get coal on the track. Most mines are sufficiently protected against squeezes during idle times. The percentage of loss in Ohio, Pennsylvania and West Virginia from this cause has been reduced since the previous study was made.

Too Many Rooms.—A peak market demand causes a tendency to drive too many rooms off the headings in order to increase production temporarily. The coal is squeezed, the roof caves and rooms are abandoned. Loss of coal from this cause has been tremendous, and becomes greater each year as more and more small mines are abandoned. The peak market during and after the war caused thousands of small mines to be opened. Hundreds of these mines are being abandoned each year, and much of this coal is lost forever. The loss of coal from this cause in the smaller mines has increased tremendously since 1922. Large companies, however, have balanced this loss somewhat by greater recovery due to more efficient mine layouts.

Thin Coal.—Coal that is slightly thinner than the average thickness of the bed frequently is left because cost of production is high. This

coal is lost when markets are good, because operators do not care to handle it. The amount of this kind of loss appears to be the same as it was nine years ago.

Lack of Planning.—Many mines lack competent engineering supervision and future development is not adequately planned. Fortunately the number of such mines is becoming less and less each year. More coal is being produced under engineering supervision than ever before. The loss of coal from this cause has decreased measurably in the last few years.

Break Lines.—Irregular and unsystematic break lines cause losses of coal. These lines are being controlled better than they were in former years, particularly in the larger mines, and the loss of coal from this cause is gradually decreasing. Idle days, fluctations of market and strikes cause delays in mining and loss of coal. The loss from these causes has increased in the past few years, although no serious strikes have occurred recently. The spasmodic market, which controls the output of many small mines, and low percentage of operating time have increased losses of this kind.

Delay in Retreating.—Retreat mining is sometimes delayed in an effort to produce coal from virgin areas at a minimum cost; losses result from squeezes and roof caves. Such losses are particularly prevalent in the small mines, which are shipping coal only during fall and winter months and are idle in the summer. Losses from this cause have been increased materially since 1922.

Machine Cutting.—The influence of mining machines upon loss of coal is a large study in itself. The usual cause of loss in machine cutting is that several inches of coal on the top and bottom of the bed is left in place. The coal which remains should be scrapped but in many mines it is left in place. Some of this coal could be classed as unavoidably lost because it is needed either to hold up the roof or to prevent the roof from air-slacking, and to make a good bottom for laying track. As a rule, coal that is cut and loaded by machine is cleaned up more thoroughly than in mines where it is mined and loaded by hand. The percentage of recovery has increased in mechanized mines since 1922.

Undesirable Constituents.—Much coal is left in place because the ash, sulfur and phosphorus content prevents it from being sold when market competition is keen. This economic problem is large in connection with the Pittsburgh bed, particularly in the Fairmont district of West Virginia and the Belmont district of Ohio. Large companies, which sell coal on various specifications, are able to load this coal separately, and although they may sell it at less than production cost, it does not constitute an entire loss. The principal loss occurs in the Pittsburgh bed in areas where a few inches of the top of the bed and several inches of the lower part of the bed are high in sulfur and ash; the loss of coal from this cause is practically the same now as it was in 1922.

Ground-hogging.—In former years, particularly in times of a good market, it was the custom of small operators to buy a few acres of coal, "ground-hog" it, mine only the thickest part of the bed and leave the thin coal and large pillars. Thousands of acres of coal have been ruined in Ohio, Pennsylvania and West Virginia in this way. In recent years this practice has become less frequent, principally because there have been no peak markets. The loss of coal from this cause is gradually decreasing.

Cost of Mining.—Large quantities of coal are lost in mines operating on a small margin of profit. When operators see that they are driving into a fault they abandon the coal rather than drive through the rock. Where a rock fault cuts out the coal there is a gradual decrease in thickness, and this thin coal is invariably abandoned. The loss of coal from this cause is practically the same as it was 10 years ago.

Handling and Preparation.—Much coal is lost in handling and preparation. Prior to 1925, this loss was principally due to inferior methods of cleaning coal; since that time cleaning methods have improved and less coal is being lost. Coal discarded on gob piles, because it is not properly separated from the impurities, has been decreasing in quantity. Mine inspectors have been attempting to prevent this practice, particularly in gaseous mines, because it constitutes a fire and explosion hazard.

Transportation.—Loss of coal in transportation in and outside the mine is not as large as it has been in former years. Mines are kept cleaner than formerly. Less coal is shot off the solid, therefore the loss of pulverized coal, by scattering over the floors and gob piles by large shots, has decreased.

Physical Condition of Coal Bed.—The physical condition of the coal bed influences the recovery. This influence seems to be almost uniform during good and bad market conditions. Thick binders or partings near the top or bottom of the coal bed are left in place and used as roof or floor. The coal above or below them is lost. Many beds have thin benches of coal on top or bottom. The removal of the parting between these benches and the main part of the bed increases the cost of mining so that much of the thin coal is left in place. Coal lost from these causes is practically the same as it was in 1922.

Bad Roof.—Unnecessarily large quantities of coal have been left on the top to hold up bad roof. Adequate timbering would eliminate much of this loss. In some districts where top coal is left in place it is supposed to be recovered when pillars are pulled. Miners neglect to pick it down and a break comes before it is recovered. The loss of top coal does not appear to be influenced to any great extent by economic conditions. There is a slight tendency to leave more coal up in order to minimize timbering and thereby reduce mining costs when the market for coal is poor. The loss of top coal has not increased since 1922 because large

mines are timbering more systematically and are making an effort to mine as much of the bed as possible.

Fine Sizes.—Before the price of coal was increased during the war, it was common practice to discard fine sizes because the market demand was entirely for lump coal. Many cleaning and sizing plants have been installed since 1922 and losses from small coal have materially decreased. These small sizes now are sold for specific purposes, such as railroad fuel, power plants and stoker fuel. Cleaning and sizing have also increased the recovery of coal by making possible and economically successful the proper separation of coal from its impurities.

Protection of Towns, Etc.—The unavoidable loss of coal in reservation of pillars under towns, railroads, streets and buildings and around oil wells and gas wells is large, particularly in the thickly populated parts of western Pennsylvania and in the oil and gas fields of Ohio, Pennsylvania and West Virginia. This unavoidable loss has increased somewhat in West Virginia in the last few years because special precautions have been taken to protect coal mines that are operating in oil and gas fields. The percentage of loss in Ohio and Pennsylvania is practically the same as it was nine years ago.

Property Lines.—In all districts large pillars are left between property lines. In the eastern part of Pennsylvania bituminous coal field, properties are usually small and the quantity of coal in property barriers is large. These property barriers are seldom recovered because no two properties are worked out at approximately the same time. The new barrier law enacted by the 1928 Pennsylvania Legislature will decrease materially the quantity of coal lost in barrier pillars. The losses in Ohio and West Virginia will remain practically the same.

Conclusions

Some definite conclusions can be drawn concerning the changes in coal losses since 1922. The development of the mechanical stoker and pulverized-fuel installations for industrial and domestic purposes has increased the demand for slack coal. Prior to the war there was little market for this size of coal except for railroad fuel. This new demand has enabled operators to sell fine coal which formerly was left in the mine or piled on the dumps. The demand for fine coal probably will increase as more stoker installations are made and as the demand increases for coal for manufacture of gas and by-product coke.

A definite consumer's attitude has crystallized in the past five years. Consumers now demand a clean coal under specifications as to size, ash and sulfur content. When market conditions are extremely competitive much coal of inferior grade cannot be mined and parts of coal beds are discarded. This coal is seldom recovered and eventually will be abandoned unless there is an unnatural stimulation of market because of

war or sudden industrial expansion. The attitude of the consumer is causing a waste of coal which cannot easily be prevented.

Operating companies have found in the past few years that successful operation depends primarily upon adequate systems of mining, proper planning of operations and reduction of costs. This strict operating supervision is particularly emphasized by larger companies and it has resulted in a decrease of avoidable losses. This decrease has taken place more rapidly in Pennsylvania and West Virginia than in Ohio. Counteracting the gain in recovery by more efficient operating methods are the losses caused by irregular operation. The markets of the past few years have been seasonal, and with the inevitable result that coal is lost from numerous causes which arise when operations are not continuous. With the inauguration of a part-time week, losses from irregular operation will again be reduced.

The public is now demanding prepared coal, both for industrial and domestic consumption. The installation of preparation equipment is resulting in a better recovery of coal in Pennsylvania and West Virginia. Coal that could not be economically prepared by hand formerly was discarded but it is now being handled successfully by preparation machines. The increase in recovery undoubtedly will increase as more preparation plants are installed.

Operators have been able to control labor adequately in the past five years and losses which resulted from labor conditions in 1922 are no longer taking place. Coal is now being mined which would have been prohibitive in cost in 1922.

Table 1 compares the losses in 1922 with those of 1931. The comparison has been made by a review of previous data, and by comparing them with supplementary data which have been gathered in the past nine years.

TABLE 1.—*Summary of Percentage of Coal Losses*

	Losses in 1922			Losses in 1931			Increase	Decrease
	Avoidable	Unavoidable	Total	Avoidable	Unavoidable	Total		
Ohio	26.0	14.0	40.0	26	14	40.0	0	0
Pennsylvania	15.5	13.1	28.6	14	13.1	27.1	1.5	0
West Virginia	10.5	12.3	22.6	9	13	22.0	0.6	0

The avoidable and unavoidable coal losses in Ohio are the same in 1931 as they were in 1922. Local conditions have changed but they have very little influence on the total quantity of coal mined. The avoidable losses in Pennsylvania have decreased from 15.5 to 14 per cent., principally because of the consolidation of numerous large mines and more

efficient operating policies. The percentage of unavoidable losses remains the same. The effect of the new barrier pillar law has not yet been felt and coal beds of the same physical and chemical character are being mined now as they were in 1922. Pennsylvania has gained 1.5 per cent. in recovery in the past nine years. Avoidable losses in West Virginia have decreased from 10.5 to 9 per cent. As in Pennsylvania, this decrease is due to more efficient operating methods. Preparation of coal also has played a large part in the decrease in West Virginia. The unavoidable losses have increased from 12.3 to 13 per cent., on account of the more effective protection of oil and gas wells and because of the attitude of consumers who demand clean coal under specifications. The total increase in coal recovery in West Virginia since 1922 is 0.6 per cent.

West Virginia leads the three states with a recovery of 78 per cent. Pennsylvania is second with a recovery of 72.9 per cent. and Ohio is third with a recovery of 60 per cent. The percentage of recovery in Ohio has not changed since 1922.

The next 10 years should see a slight increase in percentage of recovery in Ohio but probably it will not be more than 2 per cent. of the total quantity of coal mined. The recovery in Pennsylvania and West Virginia will gradually increase as experience and research in mining operations begin to yield their fruit. Installation of preparation equipment and mechanization of mines will tend to increase recovery. It is doubtful, however, whether the increase in recovery will exceed 3 per cent. in these states, because unavoidable losses are more constant than the avoidable losses and there is no economic condition in sight that will radically change the percentage of loss due to unavoidable causes.

DISCUSSION

(*Thomas G. Fear presiding*)

T. G. Fear, Fairmont, W. Va.—Mr. Sisler has given us all something to think about. West Virginia is slightly in the lead, but still too low.

In regard to the avoidable losses due to the price paid per acre for the coal, I believe that the operator who does that is wrong and is misinformed by someone in his operating department. In some fields it is possible to work the rooms and leave all the pillars, and under proper roof conditions the pillars can be pulled.

H. N. Eavenson, Pittsburgh, Pa.—That was my belief until a year or two ago. I think there are different conditions in the different parts of the state. There are companies in the southern part of the state where roof conditions are good when first mining is being done and bad when pillar work is being done, and I believe they are saving money by leaving the pillars. From a conservation viewpoint, this is entirely wrong, but if one can avoid using red ink by doing it, it is the proper thing to do. I know one or two cases where that is so. Unless the top is bad in first mining and pillar work is good, I do not believe the condition does prevail. The figures that I have seen, and I believe they are correct, show that some money actually is saved by leaving the pillar coal in. I do not think it is a general practice and do not think it is the right thing from the viewpoint of the proper use of our natural resources.

T. G. FEAR.—In one of our low coal Pennsylvania mines we are working double rooms by conveyors, recovering the pillar between the conveyors. We do not draw the 5 or 6 ft. of coal between adjoining sets of double rooms and really lose less coal by saying we will leave it in than by saying we will recover it and not getting it.

F. F. JORGENSON, Fairmont, W. Va.—How much coal loss from irregular operations can we look for before these losses will be reduced?

J. D. SISLER.—When operating two or three days a week the losses are not so great, but it is different when mines work one or two weeks and then are idle for one or two weeks. In some roof conditions it does not make any difference whatsoever.

M. W. HORGAN, Fairmont, W. Va.—The depressed coal market has a great bearing on the situation Mr. Sisler described; coal left in the pillar any length of time is not worth recovery. The coal land is being taken back by the state about as fast as it can be for unpaid taxes and the recovery is not as important as it used to be.

J. R. CAMPBELL, Scottdale, Pa.—It might be interesting to observe that according to the latest government reports for 1930 the indications are that from mechanical preparation plants, the loss is about 8 per cent.; thus for every 50,000,000 tons of mechanically prepared coal 4,000,000 tons of refuse goes out on the dump. This does not include the coal that is hand-picked. The good coal lost with the washer refuse is real dollars and cents. The average coal loss in the refuse may safely be taken at 25 per cent. of its weight, or 1,000,000 tons annually.

H. E. NOLD, Columbus, Ohio (written discussion).—Mr. Sisler says: "West Virginia leads the three states with a recovery of 78 per cent.; Pennsylvania is second with a recovery of 72.9 per cent.; and Ohio is third with a recovery of 60 per cent. The percentage of recovery in Ohio has not changed since 1922. The next 10 years should show a slight increase in recovery in Ohio, but probably it will not be more than 2 per cent. of the total quantity of coal mined."

Mr. Sisler probably is correct in his statement that the next 10 years should show a slight increase in percentage of recovery in Ohio and that probably it will not be more than 2 per cent. of the total quantity of coal mined. It may be interesting to see why the present comparatively low percentage of recovery in the Ohio coal mines probably will not increase materially during the next 10 years. The picture can be presented best by examining the conditions in the three major coal-producing districts of Ohio, followed by a few words of explanation regarding the remainder of the coal-producing areas.

In the Hocking Valley field, the physical conditions, so far as the overburden is concerned, are such that high percentages of recovery should be possible, but there are two reasons why such high recovery is not obtained and why it will not be obtained in the future. In the first place, there are no new mines of any size in this field. Only a few of the mines operating at present have any considerable acreage of solid coal yet to be developed, and in most cases the solid coal tributary to the present mines is not in large enough blocks to justify or make economically possible the radical change in mining system that would be necessary to materially increase recovery. In the second place, the Hocking Valley field is an old mining field and in a rather large proportion of the field the coal belongs to the operating or holding companies while the surface of the land is owned mainly by small farmers. In the original purchase of the coal, when the mineral ownership was separated from the surface ownership, the mining rights given to the coal owners were not sufficient to make the economical removal of pillars possible. By this I mean that the operators are liable for damage to surface caused by the caving that would follow removal of pillars. These two factors working together seem to indicate that no large increase in recovery can be

expected in this field. Perhaps two or three undeveloped areas still remain where fairly large mines may some day be developed and where it will be possible to obtain a high recovery. It is doubtful whether this will occur within the next 10 years.

The Cambridge field lies mainly in Guernsey and Noble counties. The coal being mined is the Upper Freeport, or No. 7, as it is known in Ohio. This also is an old mining field with comparatively few large areas of undeveloped coal left. Some effort is being made in this field to adopt a mining system that should increase the percentage of recovery somewhat, but it is being considerably hampered by the fact that the coal is spotty; that is, there are many faults (cut-outs or wants) which make it difficult to plan a mine system sufficiently far in advance to assure a satisfactory pillar recovery.

The Eastern Ohio field, mining principally the Pittsburgh or No. 8 seam of coal in Belmont, Harrison and Jefferson counties, is severely hampered in increasing percentage of recovery by the fact that the coal is not high enough so that a coal roof can be left in the working places on the advance. The draw slate that is exposed along the pillar edges softens and crumbles materially and is a decided deterrent to economical pillar recovery. Here, as elsewhere in the state, a considerable change in mine layout will be necessary in order to furnish enough pillars to make recovery of them worth while. Many mining men in this field assert that pillars could not be recovered even if the draw slate could be handled safely and cheaply enough, on account of the heavy bed of strong Fishpot limestone which lies from 8 to 20 ft. above this coal. The rock under this limestone and over the coal is a soft calcareous shale. They believe that this limestone could not be broken and therefore any attempt to recover pillars would inevitably result in a squeeze and consequent failure of the plan. This has been the experience wherever pillar recovery has been tried, so far as the author knows, with the present small pillars. According to the best information the writer can obtain, this limestone has actually been broken in a number of places, and at times where the break was not wanted. The writer believes that the breaking of this limestone would offer no serious difficulties if a plan of mining were adopted that would leave large pillars on the advance. Here also, as in the two aforementioned fields, there are many mines with fairly large capacity that will operate for a period of years, and yet most of them do not have solid coal in sufficiently large blocks to justify the change of system that would be necessary before any extent of pillar recovery could be attempted safely. There are still a number of rather large areas of undeveloped Pittsburgh coal in this field and it is quite possible that when these remaining areas are developed a serious attempt will be made to solve the problem of pillar removal. It seems doubtful that any material advance in the percentage of recovery will be made here in the next 10 years.

The majority of the mines in the other mining districts of Ohio are of comparatively small capacity and the properties owned or leased by the operating companies are also generally of comparatively small acreage. It does not seem likely that any material improvement in mining methods will be made by these small companies, although in a considerable number of instances, physical conditions are favorable to increased recovery.

If Mr. Sisler were to project his prophecy 20 to 30 years into the future, he might change it considerably as regards the Ohio situation. Ohio still has large areas of undeveloped coal, much of which is from 3 to 4½ ft. thick and considerable of which will have to be treated in washing or beneficiation plants before it can be marketed. It is probable that a number of new mines will be opened within the next 30 years in these undeveloped areas. It seems reasonable to expect that more modern mining methods with higher percentages of recovery will then be employed.

At another place in his paper Mr. Sisler makes the following statement: "The installation of preparation equipment has resulted in a better recovery of coal in

Pennsylvania and West Virginia. Coal that could not be prepared economically by hand was formerly discarded, but it is now being successfully handled by preparation equipment. The increase in recovery undoubtedly will increase as more preparation plants are installed."

These same forces—the benefits accruing to mining through the installation of preparation equipment—are also at work in Ohio. During the last two years four coal-washing plants have been built and placed in operation in connection with the tipples of four different Ohio mines. The writer has learned within the past few weeks that the plans for a fifth coal-washing plant are now being made and that probably it will be built in the near future.

In the Hocking Valley field the egg and nut coals, as loaded at the faces, frequently contain rather high percentages of extraneous ash, because the slate (shale) of the middle parting tends to break down in mining. This slate is mainly in the form of thin slabs, just small enough to go through the screens for 4-in. lump and large enough to remain on those for the egg and nut sizes. The operators in this field, within the past two years, have taken advantage of the shape of the pieces of this impurity and have employed what they call mechanical slate pickers. These can be installed on any shaking screen at the expense of a few hundred dollars. They are, in effect, screen plates with raised ridges in which inclined or nearly horizontal slots are placed. The thin pieces or slabs of slate pass through the slots, while the more nearly cubical pieces of coal pass over them. This simple device has been installed at a number of the tipples in the Hocking Valley district and has proved a cheap and effective way to clean these two sizes of coal. The pickers are made by a company in Pennsylvania.

Operating Organization at Mines of Consolidation Coal Co.

By A. R. Matthews,* Fairmont, W. Va.

(Fairmont Meeting, March, 1931)

This description of the organization of the Consolidation Coal Co. is intended to include only the portion that is charged with the responsibility of the actual operation of an individual mine, although it has seemed advisable to include several of the all-division personnel, who necessarily come in direct contact with the mine officials.

The Consolidation Coal Co. has a staff and line organization and, while most of the contact between these two wings is made between officers ranking above those discussed in this paper, field representatives of several of the staff departments play such an important role in the control of certain phases of activity at the mines that a true picture of the mine organization cannot be had unless these men are included.

The demands made upon the producer of large quantities of coal have certainly never been so exacting as they are today. Reductions in the number of accidents and in the cost of individual accidents has passed from the category of a desirable achievement to an unqualified necessity. Although the market price of coal today is lower than it has been in many years, consumers have never been so critical of quality. This market condition is especially severe on the producer to whom one bad shipment may mean the loss of a contract. And the battle to keep the reduction of production costs abreast with constantly dropping prices wages incessantly.

For the legitimate producer this situation demands a mine-operating organization that is comprehensive, versatile, efficient and economical; and it is in an attempt to meet these conditions that our present system has been designed. To say that this line-up is incapable of being improved or that it is applicable to all conditions would be to assume a degree of presumption which certainly is not intended here, but our system does meet the conditions which we are called upon to face today with a smoothness and thoroughness which we consider reasonably satisfactory.

Authority or Consultation

On the chart of Fig. 1 an attempt is made to show as clearly as possible the authority and responsibility of each mine official, together with the

* Division Superintendent of Mining, Consolidation Coal Co.

relationship of certain staff departments of these officials. The solid black lines are the usual lines of authority found on any organization diagram and have the same significance. The dotted lines are termed "lines of consultation" and are used to indicate contacts between individuals—contacts that are recognized and encouraged in so far as discussion of problems and consultations are concerned, but they carry with them no element of authority. They are, of course, reversible—that is, either of the men connected by one of these dotted lines may consult the other without regard to ranking positions.

Confusion between these two lines of contact—lines of authority and lines of consultation—is so common and causes so much friction and misunderstanding that the author has spent considerable time in attempting

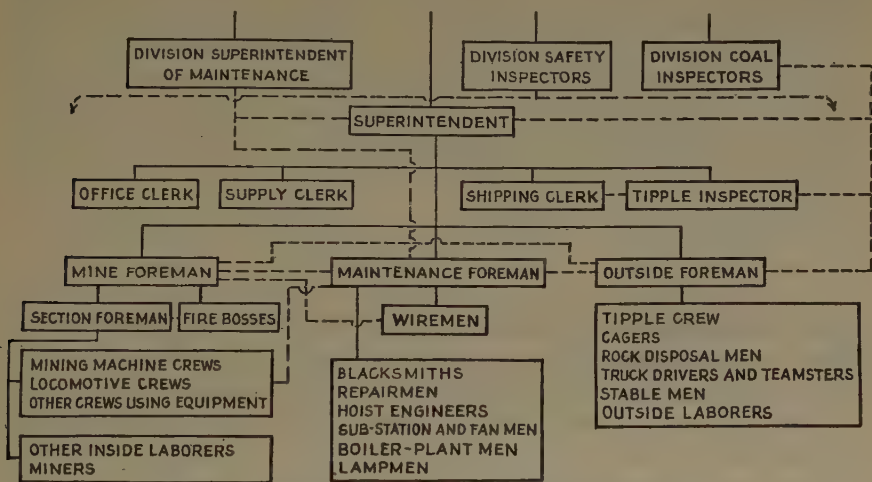

FIG. 1.—ORGANIZATION CHART, CONSOLIDATION COAL CO.

to develop the distinction between the two. The danger of trouble from this source is greatly enhanced in an organization with strong staff departments, where an unhealthy condition can quickly and easily develop. By the use of these two lines on the diagram an attempt is made to meet the first requirement of good organization; that is, clearly defined responsibility and authority for each official.

All contacts between the mine organization and the division personnel shown are by lines of consultation, although certain of these men, of course, have emergency authority. We believe that the conduct of all activities at the mine are properly the responsibility of the mine-operating organization and we do not approve of divorcing from an official any authority that normally would be exercised in discharging his duties and supervising the activities coming under his jurisdiction.

Division Personnel

The function of these members of the division personnel in connection with an individual mine is threefold:

1. To cooperate with and assist the mine officials to attain the best results possible in the particular activity in which the division representative is primarily interested.

2. To observe and report for correction any undesirable conditions or practices.

3. To note any desirable practices or conditions developed at any mine which might be applied advantageously to another.

Our mine officials have gradually come to look upon the division men as their friends, and we have an excellent spirit of cooperation between the two branches. This is due, we believe, to the fact that the system is actually functioning and the mine officials place real value on the assistance given by the division men, and accept critical reports with a constructive attitude; also, because these division men are as anxious to report unusually creditable conditions and practices as they are to report undesirable ones.

Mine Staff

The Division Safety Inspector is connected by a line of consultation to a bracket embracing the entire mine personnel, but he has the authority in an emergency to take any steps that he may deem necessary, even to closing a mine, in the protection of life and property.

The relationship to the mine personnel of the Division Superintendent of Maintenance and the Division Coal Inspectors is clearly shown and requires no comment, other than that the Coal Inspectors have the authority to condemn any car of coal or temporarily stop any particuliar working place or places when they deem such action necessary. If the mine officials do not agree that the coal from such a place or places is unfit for loading, they may protest the action and a final decision will be reached in conference.

The Superintendent is shown heading the entire organization in so far as authority goes, and being vested with this authority he is held solely responsible for the proper operation of the facilities at his disposal.

The Tipple Inspector works directly under and reports to the Superintendent. His duties are to prevent shipment of substandard coal, so far as can be determined by visual inspection, to see that the tipple is properly operated at all times, and to select cars for special consignment. He must see that all railroad cars are properly cleaned before they are loaded and must inspect the coal loaded in each car. He is vested with the authority to stop the tipple at any time that the screening or cleaning

of the coal is seriously impaired, and see that proper corrections are made. He keeps a complete record of each railroad car loaded. The writer believes that except in the case of coal shipped for metallurgical or by-product use, most rejections and complaints are based on visual inspection, and this tipple inspection is a valuable aid not only in controlling visible quality but also in checking over complaints of customers.

The distribution of the supervision under the Superintendent as between the Mine Foreman, Maintenance Foreman and Outside Foreman is not unusual. Usurpation by one of these men of authority and responsibility properly belonging to another is not tolerated. We believe that an attempt to do the work of another necessarily means neglect of one's own. Such a condition invariably creates friction and decreases the efficiency of the entire organization. These three foremen are connected by consultation lines, as we recognize that open and free contact between them promotes good feeling and harmony, and tends to increase the effectiveness and efficiency of the supervisory forces. It also tends to increase a healthy condition of rivalry between the departments. Consultations assist greatly in developing the *esprit de corps* of the organization, which is of particular value in a company operating a group of mines, as this spirit can be appealed to in promoting almost any idea that seems desirable.

The Maintenance Foreman maintains a contact through a line of consultation to all men and crews using equipment, and he is expected to keep himself informed as to the manner in which the equipment is operated and to educate the operators in the best manner of using the equipment, both to obtain the best results possible and to avoid unnecessary abuses. If he knows of an operator who wilfully abuses his equipment and refuses to accept proper instructions, the Maintenance Foreman is expected to consult with the Mine Foreman, and the Mine Foreman generally removes the man. If the Mine Foreman does not see fit to take this action, and the man continues his undesirable practice, the Maintenance Foreman may call a conference between the Superintendent, Mine Foreman and himself, and present his case. A decision will be rendered by the Superintendent after reviewing the position of the Mine Foreman and the Maintenance Foreman.

The Wiremen are shown on the diagram as being under the supervision of the Maintenance Foreman, although they are connected with the Mine Foreman by a consultation line as all their work comes under his observation and there must be a rather close contact at this point. However, they are directly responsible to the Maintenance Foreman, because we believe that this man is better fitted by experience and training to supervise the work done by this class of men, and he is held responsible for the quality of the work which they do.

Appeal from Instructions

We do not believe in the absolute authority of any one man in an organization and have established a procedure by which any official may appeal from instructions issued by his immediate superior. For example: if a Section Foreman conscientiously objects to instructions given him by the Mine Foreman, he may request that the Mine Foreman accompany him to the Superintendent for a review and appeal of the case. The Superintendent's decision is the final authority on the question. Such a review, with both officials present to defend their positions, seldom leads to antagonism or hard feelings. It is accepted as an established practice of the company.

Occasionally difficulties arise but they are always adjusted. On one occasion a Maintenance Foreman reprimanded by his Superintendent for failing to carry out certain instructions remarked that he was taking his orders from the Division Superintendent of Maintenance. The Superintendent quickly corrected this misunderstanding, and the foreman is now one of best in the company. On another occasion, a Superintendent was under fire for the excessive number of equipment failures at his plant. In defending himself against this charge, he took the position that he was not responsible for these because the maintenance of equipment came under the jurisdiction of the Division Superintendent of Maintenance. These two incidents merely illustrate unfortunate misunderstandings which arise when responsibilities of officials are not clearly defined and are chargeable to the method of inaugurating the system rather than to any inherent defects in the system itself. The cases cited were isolated ones, and do not illustrate the general reception of the scheme, which, in most cases, was installed and operated with little friction at any point.

DISCUSSION

(*Robert M. Lambie presiding*)

J. C. White, Pittsburgh, Pa.—Mr. Matthews brought out the point that organization is human and we must be careful when we work with other men. Each man is an individual and requires individual treatment. However, in a large corporation a certain unity of action and definiteness of policy is necessary and here the organization chart proves its worth. The work of a staff organization is a reflection of the man at the head and the whole organization as it is set up is the tool of the management. Our own organization is to a large degree conducted along the same manner, each man to his own job as far as that is possible. When friction exists, it has to be cleared up by the man at the head.

T. G. Fear, Fairmont, W. Va.—This system is only for one mine; we have 30 mines operating at the present time and they cannot be operated without an ironclad system. This system is used in all divisions. We also use the triangle system—if there is a question between the Division Manager and the Division Superintendent of Maintenance, the latter may appeal to the Engineer of Maintenance in Fairmont,

which inverts the triangle, then if they do not agree, the Engineer of Maintenance and I (Safety and Transportation Engineer) have a conference.

H. Emerson, New York, N. Y.—I am much pleased to see that able diagram, but it is also true that diagrams are not evolved to suit each incident without conflicting with certain universal laws, and all organization diagrams come under certain laws. The Superintendent, the Division Superintendent of Maintenance, and Safety Inspectors are not at the top, they are at the bottom; it is their responsibility to carry all that there is in the organization and that is the chief responsibility, as in the human family, those who control the family are the babies, not the parents—everything has to revolve around the baby, that is what the family is for. In industry, the whole industry revolves around the people who seemingly are at the bottom. The line executive has authority to direct and also obligation to obey the laws of the United States, or a state, or the charter of the shareholders, or the directors. The staff counselor has knowledge but no authority. He is an interpreter of the laws of the universe. A man most needed is the man who without authority and also without special knowledge knows how to coordinate all other functions and factions.

M. W. Horgan.—Where are trackmen included?

A. R. Matthews.—Trackmen are included under other inside labor.

Measuring Mine Costs and Production

By N. A. Elmslie,* Barrackville, W. Va.

(Fairmont Meeting, March, 1931)

This subject covers much ground, therefore it must be treated in a general way rather than in detail in this paper.

Personnel

To approach the measure of a mine, it is, of course, essential that the personnel be measured, and this measuring must be done in a circumspect manner and conclusions must be reached slowly. There may be one unit in the personnel of an organization that may be out of step with the remainder of the organization, not always because of lack of ability, but because of misplacement or lack of proper measurement of his capabilities. Too much time can scarcely be spent in the study or measure of the personnel; in fact, a manager or a superintendent of a mine never completes his work along this line.

Mine or Plant

Just as no two men are alike, so no two mines or plants are alike; and yet there are many similarities. Find out what the similarities are, and this will help to make clear the dissimilarities. Try to visualize details of the difficulties, not because you will have to deal with them yourself, but to consider them in later studies of general problems.

To outline all the steps to be taken in familiarizing oneself with a plant or operation would be impossible, but the following points may be of interest in any investigation.

The study of a mine is generally made backwards—that is to say, one sees the finished product first—but this is immaterial, because each step must be considered separately in any event. As a rule, there are certain heads under which the different features of mining can be placed, as follows:

Ventilation.—Ventilation is usually the most important item in power cost and safety, especially in gassy mines. Quantity of air, water gage and power consumption, and finally analysis of the return air in full return and separate splits should be considered. Type of fan and type of power unit should enter into this study because it is important to view expenditures which are going on 24 hr. a day and 365 days in the

* Superintendent, Marion Division, Bethlehem Mines Corporation.

year. Later studies may lead to possible economies. Seldom is it necessary to go further than 1000 ft. from the fan or mine opening to find the cause of high ventilation costs.

Drainage.—Like ventilation, drainage may assume important proportions, as it also is a 365-day cost. Ditching to a central basin will save many dollars.

Transportation.—This should include haulage (primary, secondary and face), hoisting and dumping, and should be considered in these units separately and collectively and as reflecting one against the other. Haulage is 90 per cent. of most mine problems. This percentage may be reduced slightly when some major problem is presented, such as timbering or bad roof conditions, but can scarcely be considered as less than 80 per cent. of the problem in any mine.

Power.—Various types of power may be used; electric alternating current or direct current of different voltages with open or permissible motors. Compressed air is used for many purposes in mining; some mines are equipped with air locomotives, hoists, drive mechanisms for conveyors or loading machines, as well as cutting machines and drills. Under the last three heads come the major features of coal mining.

Methods

Under this head must be considered mining methods, timbering methods, loading methods and coal fracture methods, whether by mechanical means or through the use of explosives. Under the head of explosives come the permissible explosive and the new carbon-dioxide type which is said to be safer than any when gas is present.

Mining systems—shortwall, longwall, room and pillar, pillar splitting or whatever system is used—should be studied. It is not always true that because coal has been worked in a certain way it is the only way, or that the method will not admit of variations. Radical changes in mining methods must be cautiously approached, however.

Production

Production in a coal mine can be considered from several angles—total production, production per unit and production per man, the latter divided into two classes where piece rates for loaders are paid. The item of production must be considered according to demand, also according to economic capacity. Too little tonnage or too much tonnage may be expensive. There is a proper tonnage for every mine, or plant, which is the economic tonnage.

Cost

The cost is a most, if not the most, important item in the series, and must be considered for marketing as well as for production. The organi-

zation can be interested in this item in many ways, even down to the loader, by creating an incentive and furnishing information on comparisons of similar factors in cost. The cost items should be split and segregated so that they can be viewed in units which can be tabulated and compared from time to time.

Revision of Methods

Before the time for revision in methods or systems, all of the foregoing items must be thoroughly digested, studies made and conclusions definitely arrived at. For example, we will take production, and all items hinge on production. Production is always limited by some one factor and no other. This one factor can be considered as the bottle-neck of the operation. When this is removed or widened sufficiently, the limiting factor will be moved to some other operation or factor. To make this clearer, let us assign percentages to various operations in a mine, the percentage of performance being based on the rate while actually working and the percentage of capacity being theoretical as follows:

Operation	Performance, Per Cent.	Capacity, Per Cent.
Loading	90	100
Face haulage	70	85
Secondary haulage	70	70
Main haulage	85	95
Hoisting	100	100
Tipple	100	100

These percentages are arrived at through studies, and from the results shown it is quite clear that secondary haulage is the item to be considered; but percentage performance alone does not indicate this, as the performance of the face haulage is governed entirely by the secondary haulage, and with this item cleared the face haulage will step up to its capacity automatically. The percentage of capacity, however, is a difficult item to determine definitely, as until the actual results are obtained it must be theoretical. On the other hand, one must not blindly apply a remedy on performance percentages alone, as it is clear that a remedy applied to the item of face haulage would bring no results whatever. Also, apparent maximum percentages on performances really are not the maximum when the next operation in the chain has been stepped up.

It is apparent that measures in coal-mine operation are the determinants of all elements of both cost and production. In other words, if the total collective operations in a mine, which are carried on at the same time, are plotted in lineal inches as a factor of time, a tape measure

can be applied and any division, or operation, that is greater than any other spells inefficiency.

To illustrate further, let 10 in. represent hoisting coal in an 8-hr. day. Then it is useless to build up haulage capacity to 15 or 20 in. On this item alone let us digress for a moment: Too many mines are overstocked with haulage equipment chiefly because of a lack of mine cars. There should be a mine car for every car of coal hoisted in a shift, or 1000 mine cars in a mine which has 1000-car dumping capacity. This allows each loader to be served promptly, even though the hoist or main haulage breaks down—shortens the day man's time and reduces haulage units, and makes possible a more uniform operation as well as a steady flow of coal from the faces. Concentration is a goal toward which all of us can safely strive. Few mines today are underdeveloped; most mines are deplorably overdeveloped. The author knows of two mines that have reduced their working sections 60 per cent. and are producing more coal than they did with the original sections working.

An organization that is not hostile to new methods is the exception, and far from desirable. The author has reached this conclusion after passing through this experience several times, and from close study of the personnel. There is more opposition from a high-class personnel than from any other, undoubtedly because of a firm conviction that they are serving the company well and doing everything possible to get results, rather than from a spirit of opposition.

When a man of this type accepts an idea, he is far more valuable than the one who passively accepts it. History tells us that Disraeli's success was due to the opposition rather than to his own followers. He learned to use the opposition to further his own ends. When we can see our opponents in the light of allies we are indeed progressing. How much real personal satisfaction there is in final achievement gained in the face of opposition!

To develop new ideas successfully in an organization, these ideas must in reality be sold, and to be a good salesman one must first sell oneself; after that the product one has to sell is easily disposed of.

Lack of perfection is what we all have to contend with, not only in ourselves but in others. Knowledge of a man's defects is of far greater value to his superior than a knowledge of his good points. The latter will take care of themselves. The manager must care for the defects, and knowledge of them will strengthen the position of the manager.

Summary

Every phase of a mining operation—personnel, mine, mining methods, production and costs—must be measured before any revision can be made, and each of these items hinges on the others. No more water can be poured through a series of funnels in a given time than will pass the

smallest of them in the same time. Money is wasted in purchasing the larger funnels unless the smaller one can be replaced by a larger one.

Stating this conclusion in another way: it is well to balance the operations in such a way as to equal the limiting factor that will not allow of revision. It cannot be too strongly stated that accurate information and the consistent use of such information in connection with detail costs and production figures is essential, and the mine superintendent who knows the actual cost of his operations, not in general but in detail, is fortified and firmly entrenched for the battle with competition.

DISCUSSION

(*Robert M. Lambie presiding*)

J. C. White, Pittsburgh, Pa.—Mr. Elmslie mentioned studies of production, mine cars and costs. I would like to ask him who makes these studies and then ask Mr. Matthews to place these men in his organization chart[1] of a large company.

N. A. Elmslie.—We have at each of our mines a man without any title. He is considered an assistant to the superintendent and his job is to follow up time studies on different operations, in conjunction with the superintendent. He does not report to the division superintendent, he reports to the man in the mines, sending copies of his reports to the division head. In making these studies we usually follow out a scheme similar to this: We study a loader—probably we take two or three loaders in each section of the mine—plot his performance lineally, so much time for laying track, so much for drilling coal, so much for loading, so much for waiting for cars, etc., and at the end of the day we have a total lineal report of what that loader does. To get an average we take a number of loaders in the different sections of the mine. This gives us a measure of the loader's production, the time he consumes in the placing of the drilling of the shot holes. Machine drilling would allow shot-firing before the man comes into his place.

This man is an engineer who has had experience in the mines and in cooperation with the superintendent walks through the mine from time to time noting details of operation and reporting possible economies. He is a medium of information for the superintendent.

Member.—How do you distinguish between first haulage and second haulage?

N. A. Elmslie.—When I speak of first haulage I mean gathering locomotives, animal haulage, etc.

J. C. White.—If the superintendent does not agree with the results of a study, what course can the man who made it take if he is reporting to the superintendent?

N. A. Elmslie.—Although these men report to the superintendent, as far as conduct is concerned, in connection with time studies, etc., a copy of their report is also sent to the division office. The division manager expresses his opinion as to whether it is a good policy to put the plan into effect. He is the court of appeals.

J. C. White.—What would be the position of these engineers in a large organization?

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[1] See page 207.

A. R. Matthews, Fairmont, W. Va.—We would bring these men in through the staff line.

H. N. Eavenson, Pittsburgh, Pa.—One point I would like to bring up is the underdevelopment and overdevelopment of mines. I think that Mr. Elmslie has put that mildly. If we knew at first what we were going to do we could get along with half the development we now have in the mines in this country and make a better job. Another point is the use of mine cars; I think there is no one item of equipment around the mines that is skimped as much as mine cars.

H. Emerson, New York, N. Y.—Not only the coal industry, but all the industrial world is overequipped; excepting Russia, where they do not yet have their equipment.

D. L. McElroy, Morgantown, W. Va.—I collected a fairly large amount of material on mine haulage and I believe I can safely say from what has been collected that 50 per cent. and probably 75 per cent. of the operators were hurting their mine efficiency by lack of mine cars. Mr. Elmslie recommends a car in use for every dump that is made. I think that statement is well founded; in fact, I know of three mines where cars to that extent were not sufficient.

A. R. Matthews.—Mr. Elmslie mentions three handlings of the mine cars from the time they leave the room until they arrive at the shaft bottom. I agree with Mr. McElroy that it is fallacious to try to determine mine-car equipment on a turnover basis. I think Mr. Elmslie's figure of one car turnover is due to the fact that he is working on the three-handlings system. What has he to offer on the matter of additional economies in haulage to offset the additional equipment required?

N. A. Elmslie.—I would not say the three-handlings system is justified unless I knew the particular mine I was talking about. At the mines I am thinking about, the three system of haulage is applicable. We run in these two mines a main line haulage, which is heavy equipment, which does not carry the car above the main lines. We try to get the sidetracks, from which the gathering locomotives handle the cars, as close to the working face as we possibly can. The secondary haulage is justified in connection with haulage from this standpoint.

When there are sidetracks more than 700 ft. from the loader's working face, each 100 ft. added means 200 ft. for each mine car, and in 50 trips a day it travels two miles more than it should have traveled. When the distance gets up to 1000 ft. the haul is doubled; if a car travels from 500 to 1000 ft. the distance is doubled to produce the same tonnage; that is where the secondary haulage is justified in not carrying heavy equipment into this territory. When mine cars are left at butt headings they have to be delivered right there, and quickly.

H. Emerson.—What is the narrowest gage of tracks?

A. R. Matthews.—The gage is 42 to 48 in.; the narrowest, 36 inches.

Member.—Can you have too many mine cars?

N. A. Elmslie.—Certainly. Depending on the distance the mine loaders are from the point of hoist, as that distance lengthened it would be necessary to add to the number of mine cars over and above the number of mine cars necessary for a day's hoisting. The number of mine cars might be increased to figures that would sound ridiculous in this matter, but just speaking offhand I say 1000 cars for 1000-car hoist.

Suppose we use 1000 cars as the basis of estimate; loading capacity 240 cars per hour loaded by 180 loaders, each loader loading 6 cars in a day. This gives a rate of 180 cars every 45 min., and demands 60 cars more than the number of loaders each hour of loading.

If the hoist capacity is 150 per hour, there are 90 loaded cars per hour in excess of the hoist capacity. These excess loads must be stored while loading is in progress in order to keep the loading capacity. The loaders will produce 1080 cars in 4½ hr., so that 4½ × 90 loads, or 415 loaded cars, will be accumulated, which the hoist cannot handle in the loading period.

Stating this in another way: Suppose we have 180 loads on hand at the beginning of hoist in the morning. These loads can be hoisted in 1 hr. 12 min. No coal from the day's loading can reach the hoist for at least 2 hr. Therefore there should be 300 loads standing at the beginning of hoist, 180 empties in the loaders' working places, 180 on loaders' sidetrack and 180 on section sidetracks. Finally, there should be 180 empties on main line sidetracks, or a total of 1020 cars. The 415 loads which accumulate during the 4½-hr. period of loading mean that 435 of the total of 1000 must be turned over twice in the 8 hr. It would seem that 1435 cars would be desirable, plus the number of cars which were dead for repairs, slate, material, etc., or 1500 cars.

If the hoist capacity is the same as the loaders' capacity, which is desirable, this would change the picture and allow the following:

480 loads standing for first 2 hr. hoisting
180 empties in loaders' places
180 empties on loaders' side tracks
180 empties on section side tracks
180 empties on main line side tracks

1,200 plus dead cars.

If the hoist capacity is greater than loaders' capacity, or 300 per hour, this would again change to:

600 loads standing for first 2 hr. hoisting
720 empties
200 extra loads standing to keep hoist moving full time

1,520

The latter shows the inefficiency of excess hoist capacity, while the balance of hoist and loaders indicates efficiency. Part of the inefficiency of low hoist capacity must be overcome by excess mine cars and by starting to hoist coal before the loaders begin loading, if enough standing loads are on hand to keep hoist going for at least 2 hr., plus the time between start of hoist and start of loading.

A. R. Matthews.—Mr. Elmslie takes the position that handling the mine cars three times from the first handling to the bottom speeds up the haulage. I think it slows it up.

J. J. Forbes, Pittsburgh, Pa.—I wonder how Mr. Elmslie makes time studies in connection with safety.

N. A. Elmslie.—It would be ridiculous to say that in this time and age we did not consider safety the paramount feature. Safety appears to us largely a matter of supervision and it is developed in connection with systematic methods of timbering in pillar work. Of course, in all of our mines we have standards of timbering for each particular case of pillaring or development.

Relation between Mine Performance and Mine Cars

By D. L. McElroy,* Morgantown, W. Va.

(Fairmont Meeting, March, 1931)

It is too broad a statement to say that the mine car is the most important unit in a haulage system, but almost every mining man will admit that it is one of the most important. The mine car is to the mine haulage system what the railroad car is to the railroads; the main purpose in each instance is to move the greatest number of ton-miles of material at the least cost.

The data presented in this paper are taken from a study of mine haulage conducted by the School of Mines of West Virginia University at 42 mines in 8 coal fields of West Virginia. The study was made possible by a graduate fellowship of the School of Mines and the cooperation of the mining companies of the state.

Three important factors in the use of mine cars in West Virginia are: (1) the number of mine cars in use per loader; (2) the capacity of mine cars; (3) the distribution of mine cars. ("Loaders" in this paper refers to all men loading coal, whether manually or mechanically.)

During the World War many mine managers were annoyed by the reduction of efficiency and production caused by railroad-car shortages. Today relatively little complaint is heard about a shortage of mine cars, yet possibly 50 to 75 per cent. of the mines in West Virginia are suffering from this cause of reduced production and increased costs. The railroads have solved their car shortage by placing in service more cars of larger capacity and better design. Many coal mines in West Virginia should solve a part of their cost problems by the same simple procedure. Such a procedure requires an initial outlay of capital, but there is little doubt that this will be repaid, with a good return on the investment.

Determining Number of Cars Needed

There can be no mathematical rule to determine the number of cars which should be in use per loader in a coal mine, because numerous factors such as concentration of mining, system of haulage, car capacity, speed of haulage, distance of working places from tipple, and so forth, affect materially the number of cars in use. These factors in turn are more or less determined by height of bed, character of roof and coal,

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grades and age of the mine, and the effect of these factors on the number of mine cars needed seldom can be accurately measured. The coal mine and haulage systems should be carefully studied and the results of this study should be the guide in deciding upon the number of cars. Also, at regular intervals the haulage should be checked, as ventilation is checked, to see whether conditions have arisen that require changes in the number or distribution of the cars.

Numerous mines have had their car needs determined by rule of thumb or guess, and the guess more often than not is too low. No industrial organization wants dead investments on hand, but it is much better to have a few more mine cars than are needed. The actual number needed is too indefinite to calculate too closely, and the extra cars act as shop cars, care for increased tonnage demands, and give one of the cheapest forms of mine storage.

A tendency in mine-car design which has received a great deal of attention in the last several years is increased capacity. This has received considerable attention in planning new coal operations and where mechanical loading has been started. The use of a car of greater capacity has had a purely economical basis and has proved to be more efficient. The design of mine cars now tends to a longer, wider but lower car of increased capacity, because of the following advantages:

1. Less work required in loading due to decreased distance of lift.
2. A more stable car due to lower center of gravity.
3. Same tonnage at slower speed of travel or greater tonnage at same speed of travel.
4. Less investment in car furnishing a given total capacity.
5. Less confusion and fewer delays because there are fewer movements per ton on haulageways and in working places.

The principal disadvantages, which have been overcome by a better standard track construction, are that it is more difficult to rerail wrecked cars and that the longer wheelbase requires longer curve radii.

Many mines in West Virginia are limited to some extent by bed thickness, roof conditions and types of equipment used to handle cars, but a fair number of these mines are not using to a full extent the advantages they have to increase car capacity. However, there is a noticeable tendency, especially in beds of 60 in. thick, or less, to increase car capacity by better design.

Distribution of Cars

To obtain the maximum return from an investment in mine cars they must be placed when and where they are needed. The duty of obtaining efficient haulage rests with the operating officials in properly distributing haulage equipment of sufficient quantity and good design.

The importance of having mine cars well distributed is realized at many coal mines in West Virginia, as shown by the employment of men to do nothing except to regulate the distribution of haulage equipment. These men, generally called dispatchers, have full charge and responsibility for the efficient handling of haulage operations. The statement has been made that wheels were placed on mine cars and locomotives so that they might be moved, and when they are not moving they are making no return on the money invested in them. On the basis of this statement the duty of the person in charge of haulage is to keep mine cars moving as much of the time as possible. The time required for loading and unloading is the only time when cars should be idle, and this should be reduced to a minimum compatible with economy and safety. Cars lying on sidetracks or elsewhere, unless acting as storage and not needed for regular production, should be eliminated.

Relation of Tonnage to Efficiency of Haulage

Of the 42 mines studied, 10 of the 12 mines producing 2000 tons or more per shift and 8 of the 30 mines producing less than 2000 tons per shift employed a dispatcher who had charge of all haulage. The 18 mines that employed a dispatcher used 61 main-line locomotives—an average of 3.39 locomotives per mine. The 24 mines that did not employ a dispatcher used 51 main-line locomotives—an average of 1.73 locomotives per mine. Table 1 gives the performance records of various units under dispatchers' orders and those not under dispatchers' orders. This table shows that more efficient distribution of mine cars can be accomplished by dispatching.

Table 1.—*Comparison of Results with and without Dispatchers*

Condition	Tons per Shift per Main-line Locomotive	Ton-miles per Shift per Main-line Locomotive	Tons per Shift per Gathering Locomotive	Tons per Shift per Unit of Stock
Dispatcher employed	678	1217	199	109
No dispatcher employed	614	846	155	72
Increase at mine employing dispatcher	64	371	44	34

If we assume such factors as thickness of bed, character of coal, depth of cut, width of place and quality of loading to be the same for two mines, the one with the more efficient haulage will have the greater tonnage per loader, because the more efficient and better equipped haulage system will replace loads with empties at the face and empties with loads at the tipple faster and more regularly. For this reason the daily

tonnage per loader is considered by many mining men to be one of the best measures of the efficiency of a haulage system.

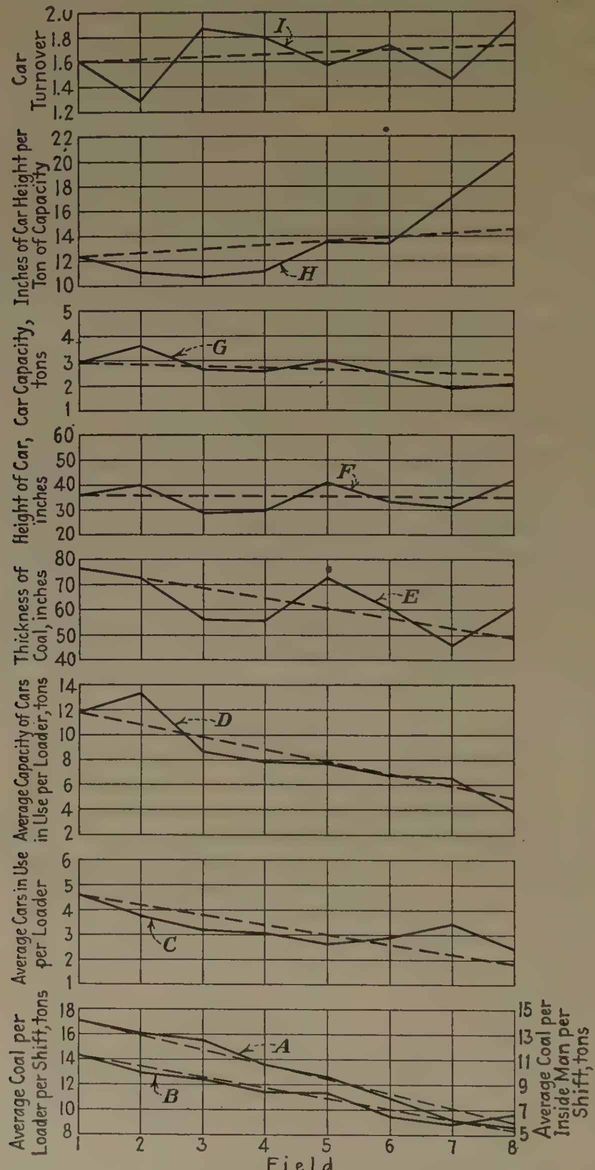

FIG. 1.—FACTORS AFFECTING TONNAGE PER LOADER.

Table 2 gives some of the factors that affect the tonnage per loader. Everything except thickness of bed and tonnage per inside man deals in some manner with mine cars. The fields visited in the study are arranged

in descending order of tonnage per loader. The data for each field are the averages for the mines visited. The tonnage per inside man is given because by including the company or day men the conditions due to long hauls, bad roof and so forth, which require extra labor, are shown. Fig. 1 indicates that the many varying natural conditions do not change greatly the trend of the tonnage per inside man from that of tonnage per loader.

The data of Table 2 are given in graphic form in Fig. 1, to show the trend and effect of the various factors at one inspection. The dotted lines show the general trend of each curve. The most noticeable and important fact brought out is the close conformity in the shapes of the curves *A*, *B*, *C* and *D*. The number of cars in use per loader and car capacity are reflected in capacity of cars in use per loader, which had greater effect on the tonnage per loader than any other factors, according to the data collected. The reason that tonnage per loader was not affected to a great extent in fields 6 and 7 by the number of cars in use—which rises while the tonnage per loader falls—is the decreased height of bed, reflected in car capacity (shown by curves *E* and *F*).

TABLE 2.—*Factors Affecting Tonnage per Loader*

Field	A[a] Tons per Loader per Shift	B Tons per Inside Man per Shift	C Cars in Use per Loader	D Capacity of Cars in Use per Loader, Tons	E Thickness of Bed, In.	F Height of Cars, In.	G Car Capacity, Tons	H Car Height per Ton of Coal Capacity, In.	I Car Turnover
1	17.10	11.29	4.61	11.82	76.8	36.2	2.90	12.47	1.61
2	16.04	9.99	3.75	13.27	73.2	40.0	3.60	11.10	1.28
3	15.55	9.45	3.23	8.68	56.4	29.0	2.70	10.73	1.86
4	13.50	8.42	3.08	7.90	56.0	29.2	2.62	11.14	1.80
5	12.57	8.31	2.64	7.76	72.6	40.4	2.98	13.55	1.57
6	10.90	6.46	2.88	6.74	60.8	32.8	2.43	13.49	1.73
7	9.12	5.88	3.38	6.51	46.6	31.0	1.82	17.01	1.46
8	8.55	6.58	2.38	4.82	61.0	42.0	2.03	20.68	1.92
Avg.	12.57	8.21	3.07	8.31	62.8	35.08	2.61	13.77	1.67

[a] Letters refer to curves on Fig. 1.

It will be noted that Curve *D* (capacity of cars in use per loader) is followed by that of curve *A* (tonnage per loader), except in field 2. The reason for this increase in car capacity (curve *G*) is shown in height of bed (curve *E*) which is reflected in the height of car (curve *F*). The reason that the tonnage per loader does not also increase lies in the long hauls at the mines of this field. Long hauls require more cars or greater capacity, which is somewhat reflected in the low car turnover (curve *I*). The average distance from face to tipple at the mines of this field was over one-third greater than the average of the other fields.

The number of cars needed at a mine depends on various factors, such as concentration, length of hauls, systems of mining and haulage, speed of haulage and the capacity of the cars. The curves show that the number of cars in use have almost a direct effect on the tonnage per loader and that the capacity of the cars in use per loader has even a greater effect.

Relation of Car Capacity to Tonnage

Car capacity, therefore, appears to be an important factor in determining the tonnage per loader, because it is the combination of the number of cars in use and their capacity which has the greatest effect. Curve *G* gives the car capacity in tons for each field. Curves *E*, *F* and *H* show the factors that affect or are reflected by car capacity.

It is generally agreed that the greatest car capacity economically possible should be used. The greatest car capacity found in the study was 4 tons and the smallest was 1.5 tons. The average height of bed at which cars of 4 tons capacity were used was 78 in., the maximum being 84 in. and the minimum 66 in. Two mines used cars of 1.5 tons capacity and had bed thicknesses of 48 and 46 in. The thinnest bed encountered was 42 in. and the average car capacity at this mine was 1.7 tons, although a few new cars of 2.5 tons capacity were in use. It is an important fact that height of the 2.5-ton car was 6 in. lower than that of the smallest car.

Experience leads the author to conclude that the greatest height a man can shovel with efficiency is 48 in., and for best results it should not exceed about 36 in. The height to which the coal is raised should be kept low in both manual and mechanical loading. Regardless of the kind of power used, manual or mechanical, every inch of additional lift in loading requires more power, which is certain to be reflected at some place in the cost sheet.

Let it be assumed that 48 in. is the maximum height of car above rail for efficient loading, and allow 18 in. for minimum clearance between car and roof. Then 66 in. of bed thickness will be the least possible without change in the height of the mine car. The highest car encountered in the study was 46 in., which had a capacity of 4 tons, used in a bed 84 in. thick. The lowest car was 24 in. high; it had a capacity of 2.5 tons and was used in a bed 42 in. thick.

Relation of Bed Thickness and Height of Cars

The average clearance between car top and roof of the mines studied was 27.72 in. The average height of car above rail was 35.08 in. and the average height of bed was 62.8 in. Both of these figures are shown in Table 2. The average clearance between car and roof for the mines having less than 66 in. of bed thickness was 18 in. less than that of the mines of over 66 in. bed. This fact substantiates the conclusion that less than

66 in. (somewhere between 60 and 70 in.) of bed thickness affects the height of cars. A study of curves *E* and *F* (bed thickness and height of car) bears out this statement as reflected in the conformity of the curves. In fact, the unconformity occurs between fields 1 and 2, both of which have bed thickness of over 70 in. The close relation between height of cars and bed thickness is well illustrated in curves *E* and *F*.

Also there is a close relation between the height of cars and car capacity (curves *F* and *G*). Although the curves *F* and *G* have the same trend, the effect of car height on capacity is not so sharp or determined as that of bed thickness on height of car (curves *E* and *F*). The reason for this is that as car height is cut down by bed thickness there is a tendency to maintain capacity by using a wider and longer car built closer to the rail. The facts brought out by curves *E*, *F* and *G* would be expected after consideration, but perhaps not so uniform as shown by these 42 mines.

Curve *H* shows the inches of car height per ton of car capacity, for each field. A low figure here indicates the use of the new car of low, wide and long design. It has been stated that when bed thickness becomes less than about 66 in. the car height is affected. Therefore, because 27 of the mines visited are in this category, the factor shown in curve *H* becomes important. More important still is the fact that, in general, as the tonnage per loader decreases the value of the factor given in curve *H* increases. Although, on account of varying natural conditions, it is not altogether correct to name a minimum value for the capacity of cars in use per loader, after this factor becomes less than 8 tons, the tonnage per loader decreases slightly faster and the inches of car height per ton increases rapidly. These facts stand out despite irregular curves of thickness of bed *E*, of car height *F* and car capacity *G*. This fact indicates that all of the factors and the number of cars in use are reflected in the capacity of cars in use per loader. The tonnage per loader follows curve *D* (capacity of car in use per loader) more closely than any other and becomes more pronounced when the value of curve *D* becomes less than 8 tons. It will be remembered that the tonnage per loader was taken as the measure of haulage efficiency for this discussion.

Car Turnover

The average car turnover for each field is given in curve *I*. Car turnover has been regarded by some persons as the measure of haulage efficiency. The data of this study as summarized in Table 2 and Fig. 1 prove that car turnover in two haulage systems is not comparable unless the systems are identical; in fact, curve *I* indicates that, in general, as the turnover increases, the tonnage per loader decreases. Probably this is due to the fact that a mine that is short of cars has three alternatives to improve the situation; namely, purchase additional cars, purchase cars of increased capacity or increase the turnover. This statement

is based on the assumption that car shortage is the only major difficulty with the haulage system. As an increase in the turnover requires little or no investment (although it will probably be strongly reflected in the power cost) this method is resorted to for bettering the situation.

The data collected indicate that if car turnover is high, considering length of haul, and other conditions, instead of finding an efficient haulage system, generally the opposite will be true, on account of shortage of mine cars.

Table 2 and Fig. 1 deal with two of the subjects to be discussed; namely, the number of cars in use per loader and car capacity per loader.

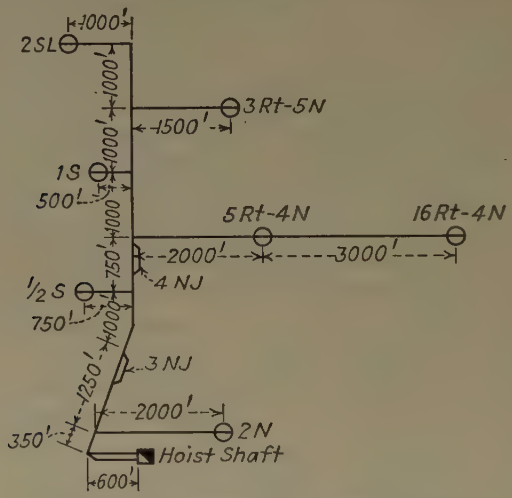

FIG. 2.—OUTLINE OF HAULAGE SYSTEM.

The remaining subject, the distribution of mine cars, is very important as it has considerable effect on the two preceding topics.

VALUE OF TIME STUDIES

In any work or process it is difficult to determine when the units are functioning properly and efficiently unless their operations are separately and collectively known and studied. The form of some dispatching sheets is such that sufficient information is collected to be practically a time study of the haulage. To illustrate the way in which the complete operations can be detailed by a time study, such a study will be discussed briefly. By such a study or complete dispatching record the distribution of cars may be made most efficient.

This time study was made of the main-line haulage system and shaft-bottom performance of a mine producing 3000 tons per shift. The main-line units consisted of one 25-ton and two 15-ton trolley locomotives. The coal was hoisted on a cage hoist operated by steam. The haulage

layout of the mine is shown in Fig. 2. On the shift on which the study was made, 1056 cars of 3.2 tons capacity were dumped. This mine averaged about 20 tons per shift per loader and about 11 tons per shift per inside man.

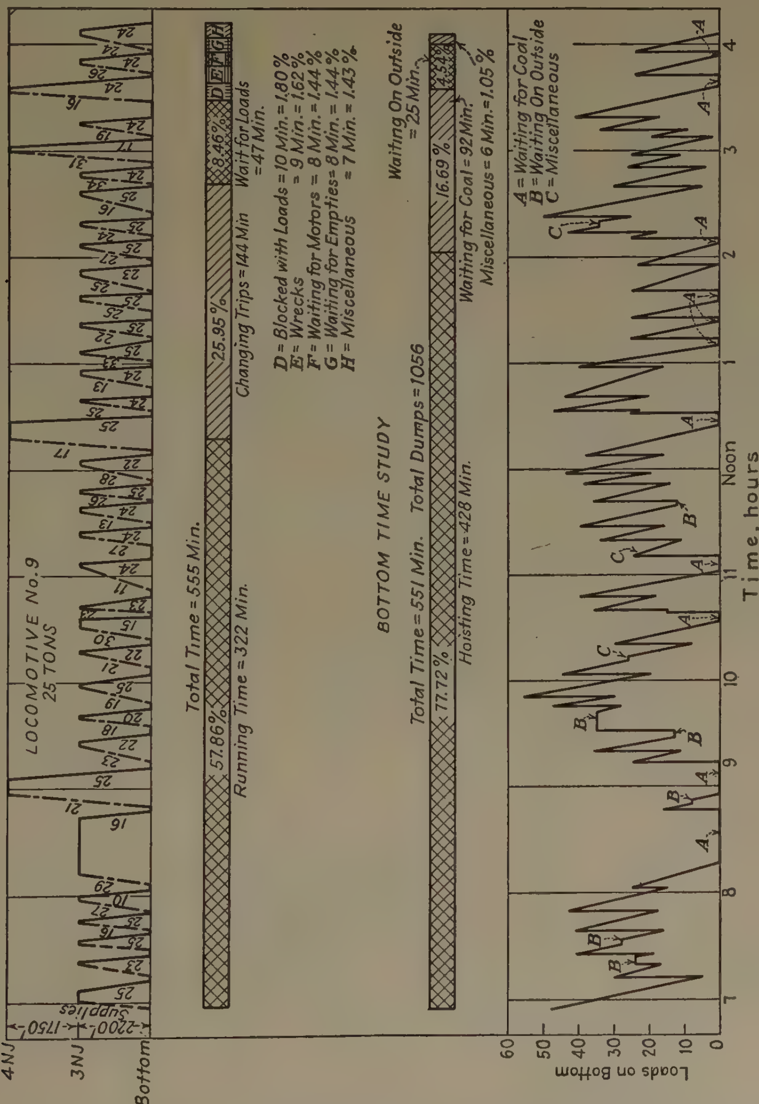

FIG. 3.—MAIN-LINE AND BOTTOM TIME STUDIES.

The time study was conducted by having the movements of each locomotive tabulated on the form shown in Table 3. In addition, men were placed as checkers at 3NJ and at the bottom; the man at the bottom also timed the bottom performance. At the completion of the study all data were collected and plotted in the form shown in Figs. 3 and 4.

TABLE 3.—*Time Study of Locomotive* 9

Place Left	Time Left	Place Arrived	Time Arrived	Cars Hauled		Remarks
				Loads	Empties	
Bottom	8:07	3NJ	8:12		29	
3NJ	8:44	Bottom	8:48	16		Waited 29 min. for loads
Bottom	8:50	4NJ	8:58		21	
4NJ	9:06	Bottom	9:12	25		Waited 5 min. for loads
Bottom	9:16	3NJ	9:25		23	Run slow: locomotive ahead
3NJ	9:27	Bottom	9:31	22		

Locomotive 9, a 25-ton unit, made scheduled trips between 3NJ and the bottom and had a trip limit of 25 loads. Fig. 3 shows that with the exception of four trips to 4NJ, this schedule was maintained. The two remaining main-line locomotives 1 and 2 were scheduled to haul from the gathering sidetracks to 3NJ. Fig. 4, however, shows that 10 trips were made to the shaft bottom and 5 trips to 4NJ by these two locomotives. In addition to these locomotives, two gathering locomotives also placed loads at 3NJ and 4NJ. All units worked under orders of a dispatcher who could change the regular schedule.

The movements of locomotive 9 and loaded cars on the shaft bottom are shown by Fig. 3. The time in hours is given as the horizontal divisions, the figures representing 7 a. m. and so on to 4 p. m. The time in hours is the ordinate for both the locomotive and bottom operations. The starting and stopping time, as well as the duration of each operation, is shown by such charts as Figs. 3 and 4. The vertical component of the bottom study is the number of loads on the bottom. For example: at 6:52 a. m. hoisting was started with 48 loads on the bottom and continued until only 5 loads remained at 7:13 a. m. At this time, locomotive 9 arrived with 25 loads from 3NJ (see chart of locomotive 9 performance at top of Fig. 3). The slope of the bottom chart gives hoisting speed, as from 6:52 a. m. to 7:13 a. m., 43 loads were hoisted—an average hoist of 1.4 cars per minute. The columnar chart directly above that showing operation of the bottom gives the time distribution of the bottom working shift in hoisting time and the various delays.

The movements of locomotive 9 are charted at the top of Fig. 3. The ordinates are time in hours, shown at bottom of Fig. 3, and the abscissas are the points visited by locomotive 9; that is, 3NJ, 4NJ and shaft bottom. The dotted lines represent trips with no cars hauled, the dot-and-dash lines represent empty-car trips and the solid lines represent

loaded-car trips. The numbers on the lines show the number of cars in the trip. Thus, at 8:50 a. m. locomotive 9 left the shaft bottom with 21 empty cars, arriving at 4NJ at 8:58 a. m. The locomotive was at 4NJ from 8:58 until 9:06 a. m., changing trips and waiting 6 min. for loads. At 9:06 a. m. the locomotive left 4NJ with 25 loads, arriving at the bottom at 9:12. Directly below the movement chart is shown a distribution

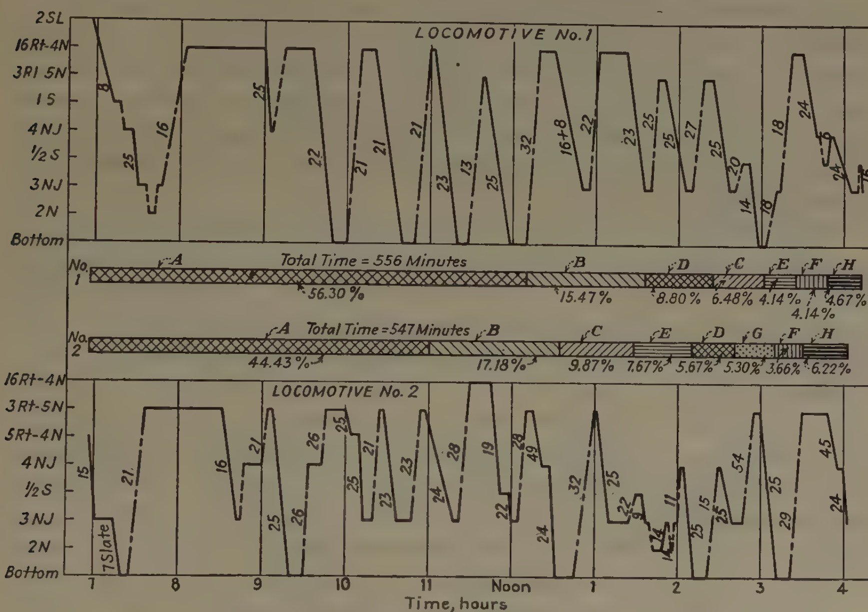

FIG. 4.—MAIN-LINE TIME STUDY OF TWO 15-TON LOCOMOTIVES.

A. Running time.
B. Waiting for loads.
C. Changing trips.
D. Waiting for gathering locomotives.
E. Waiting for locomotive 9.
F. Waiting for empties.
G. Stopping to pick up cars.
H. Miscellaneous (power off, wrecks and other causes).

chart of the locomotive's time as to running time, time of changing trips, waiting for loads and other movements.

The movements and time distribution of locomotives 1 and 2 are given in the same manner in Fig. 4. By placing Figs. 3 and 4 together, the main haulage and bottom performance for a shift can be inspected as a whole.

Conclusions from Time Study

1. Locomotive 9 ran with much greater regularity than locomotives 1 and 2 because it had a more uniform and concentrated haul. The average working time of the three locomotives was distributed as follows: running time, 52.86 per cent.; changing trips, 14.10; waiting for loads, 13.70; miscellaneous, 19.34.

Because locomotive 9 made so many trips, requiring changing time each trip, the time for changing trips was 25.95 per cent. of its total time. Including the time of changing trips as part of actual working time, as it must be, the following percentages of working time are given: Locomotive 9, 83.81 per cent.; locomotive 1, 62.78; locomotive 2, 54.30. As would be expected, these percentages show locomotive 9 to be the most efficient.

2. Locomotive 9 traveled a total distance of 30.98 miles, which was an average of 3.35 miles an hour, including full shift, or 5.81 miles an hour actual running time. The average time of a round trip to 4NJ was 16.5 min. and to 3NJ 16.3 min. However, this is a poor comparison because only 4 of the 34 round trips were to 4NJ. The average size of trip was 20 cars for both loads and empties, and should be brought closer to the limit of 25 cars for loads.

3. Locomotive 1 traveled a total distance of 31.58 miles, which was an average speed of 3.41 miles per hour for the full shift, or 6.05 miles per hour of runing time. Although this locomotive was scheduled to run only from sidetracks to 3NJ, it made five trips to the bottom. This locomotive made a nonproducing round trip from 3NJ to 2N between 7:36 and 7:48 a. m. (Fig. 4).

4. Locomotive 2 traveled 30.07 miles, which was an average speed of 3.29 miles per hour of working time, or 7.42 miles per hour of running time. This locomotive was scheduled only to haul between the gathering sidetracks and 3NJ, but made five trips to the shaft bottom. Two round trips were made to 2N from 3NJ between 1:40 and 1:58 p. m., and only placed one empty trip and pulled one loaded trip.

5. The main-line locomotives and the shaft bottom had time losses caused by waiting for coal during the early part of the shift. This fact is easily seen in Figs. 3 and 4 and in Table 4. This delay averaged about

TABLE 4.—*Time Losses of Three Locomotives*

Unit	Hour	Time Lost, Min.
Bottom	8:17 to 8:47	30
Locomotive 9	8:12 to 8:44	32
Locomotive 1	8:14 to 9:00	46
Locomotive 2	7:36 to 8:31	55
Average		41

¾ hr. for each unit, and affected 11 men of this study, which amounts to 8.25 man-hours of lost time. In addition, of course, were the gathering crews and all of the idle equipment. The cause of this delay is seen in the number of cars in use per loader for this mine, which was almost four—an insufficient number of mine cars. This mine worked a night

shift of the same capacity as the day shift, and with the large amount of stock coal standing at the beginning of the shift, and considering the length of hauls, enough cars were not in use. At first glance this mine appears to have enough cars, but because of a large night shift (an unusual condition) and long hauls it does not have a sufficient number.

6. Three times during the shift locomotive 9 waited for coal at 3NJ when locomotives 1 and 2 could have been met at 4NJ (Figs. 3 and 4).

7. At five different times during the shift the dispatcher had two locomotives on the bottom at the same time. As the bottom would not clear *one* locomotive if more than 10 loads were between the crossover and the shaft, such a condition each time caused time losses.

8. Thirty of the 41 trips placed on the bottom arrived while loads were still present to be caged. This indicates that the bottom was working nearly at capacity. There were only five lulls in the rate of arrival of trips: at 8:16 to 9:14, 10:30 to 11:10, 12:24 to 12:32, 1:10 to 2:10 and 3:36 to 4:00 (Figs. 3 and 4).

9. The average hoisting speed for the shift was 2.47 cars per minute, but for short intervals (Fig. 3 from 12:33 to 12:41) as many as 3:25 cars were hoisted per minute.

This discussion indicates that the entire haulage or even the entire operation of a mine could be plotted in somewhat the same manner as Figs. 3 and 4. Such a chart would give the picture of the entire operation separately and collectively. The detailed type of dispatcher's form, as used by some companies, practically collects these data on haulage each shift. Also, some mines use automatic hoist-recording charts which give a complete record of the hoisting operation. It is even not beyond reason to place recording watt-hour meters on equipment and thus record also the power consumption, which could be plotted with a chart such as Fig. 3 and a profile of the haulageway to study the effect of grades.

DISCUSSION

(*Robert M. Lambie presiding*)

T. G. Fear, Fairmont, W. Va.—In one case where the mine cars hold 3.2 tons, a total of 395 cars in operation including shop cars, and handling 2800 tons of coal a day with 395 cars is an average of 2.2 turnover. I do not advocate a 1-car turnover, as on this basis we would have to spend $90,000 for mine cars, the interest and carrying charges amounting to $10,800 per year and at the end of the year this item would be on the cost sheet. Efficient car turnover at the mine denotes efficiency, and it is a good way to save money.

D. L. McElroy.—I believe, particularly in the latter part of the paper, that I mentioned the fact that car turnover, although it cannot be compared for two mines, may be a desirable feature of a certain one. Car turnover may not be 2½, but in some mines if 2½ is possible, considering conditions with safety, etc., 2½ should

be used. Car turnover is hard to set definitely until conditions have been studied and until you know the cars you need in use.

N. A. Elmslie, Barrackville, W. Va.—Is it a shaft operation?

T. G. Fear.—Yes.

N. A. Elmslie.—What is the hoisting capacity?

T. G. Fear.—Three cars a minute.

N. A. Elmslie.—How many cars do you hoist in 8 hours?

T. G. Fear.—An average of 900.

N. A. Elmslie.—In other words, you could hoist the tonnage in 5 hr. if you had enough mine cars.

T. G. Fear.—I have always thought that any mine with less than 1½ car turnover was not efficient. When you go below 1½ car turnover you must do something to improve the situation. One thing that can be done is to get main-line locomotives large enough to haul the cars safely.

Without loading machines how could you load that tonnage in 5 hr.? The men cannot load it so quickly.

N. A. Elmslie.—Assuming, as you say, that you can hoist 180 cars an hour; if you cannot load 180 cars an hour, your mine is not efficient.

T. G. Fear.—Hoisting capacity has nothing to do with it.

N. A. Elmslie.—In measuring mine efficiency, and in order to produce maximum efficiency of the mine, I recommended that a car be in service for each car that was loaded.

T. G. Fear.—The men are averaging 17 to 18 tons per man, and if they were union men we could not send them home after 5-hr. work.

N. A. Elmslie.—I cannot say that the mine you are operating is not efficient nor whether it is beyond the limitation placed on it. Naturally, you cannot apply measures to operations as they are operated in West Virginia today, where less than capacity tonnages are required.

S. A. Taylor, Pittsburgh, Pa.—I have listened with interest to the discussion as to whether the large or small mines are more economical. The whole question is in having a well balanced mine; the question of size has nothing to do with it.

F. S. Follansbee, Pittsburgh, Pa. (written discussion).—The ideal number of cars for a coal mine using hand loading would be (1) a car or cars in each working place; (2) a trip of cars attached to each gathering motor. A trip attached to each main line motor; (3) an empty trip at the tipple; (4) cars at the slate dump, if there is one, and a few cars for shop and material.

Take a 3000-ton mine equipped on this basis. The loaders average 15 tons per man and work one man to the place, using 200 working places. The cars hold 3 tons of coal. The gathering motors haul 62 cars per day, or 186 tons per motor, requiring 16 motors, placing 10 cars on each round. The main-line motors haul 50-car trips and three motors are required.

Cars Required:

Working places	200
Gathering motors	160
Main line motors	150
At tipple	50
Refuse, shop and material	25
Total Cars	585

This would give a 1.71 car turnover. The same mine working two loaders to the place would have just one-half of the working places and require 100 fewer cars, giving a 2.06 car turnover.

This mine working three men to the place, where cars are delivered one at a time, would require 133 fewer cars than with one man to the place. If the working places were angle face rooms and open-end pillars, where three cars could be delivered at one time to the three men, the cars required per working place would be the same as with one man to the place, but as the gathering motors, delivering three cars at a time instead of one car, should gather considerably more coal, fewer motors would be required and a saving in cars would be made on this item.

The length of main-line haul in the above mine requires three main-line motors. If the haul were shorter and required two motors, 50 cars less would be needed. If the haul were greater and required four motors, 50 additional cars would be needed.

The tendency toward cars of greater capacity should be carried out in thick seams as well as thin. The low-slung car with the bottom as close as practicable to the rail can be made as high as the seam permits without exceeding the economical shoveling height of 35 to 40 in. Of course, with pit-car loaders or mechanical loading the height should be increased.

The chief advantage of large cars is increasing tonnage per loader and tons per gathering locomotive. A locomotive will gather sixty 4-ton cars as easily as sixty 2-ton cars. The large car will also make it possible to work more men in the narrow places, giving more concentration. Finally, where there are bottle necks such as shafts, capable of making a given number of hoists per minute, the capacity of the mine will be increased by a larger car.

So far as helping the stability of the car by lowering the center of gravity, I believe there are almost no cases where a coal-mine car has been top heavy. On the main haulage, owing to grades, the motors usually are limited to a certain number of cars per trip. This represents so many tons. If larger cars are installed and the same motors are used, the same total tonnage per trip probably will be hauled.

While the cost of the large car per cubic foot of capacity is less than that of the small car, the total investment on the basis of car distribution outlined above will be greater.

In other words, when there is no car in the working place, the man is usually needing one. When there are no cars attached to a locomotive, it is waiting for them. The same number of cars, large or small, are required except where the large car gives concentration; that is, allows doubling up in working places where the same tonnage per trip but fewer cars are handled by main-line motors, and where fewer gathering motors are needed, since by getting the same number of cars per day they haul more coal.

The wheel base need not be considered an objection, as the mine-car wheel base is seldom as great as that of the gathering motor. A car 14 ft. 0 in. over all, 7 ft. 0 in. wide inside, 42-in. gage is satisfactory on a 42-in. wheel base.

Distribution of cars by a dispatcher is essential to efficient haulage. The dispatcher is also a forwarding agent for messages, etc.

As stated before, while the bed thickness limits the height of cars, the bed thickness should not affect the width or length. The man with the thick seam should take advantage of latest car design to get the maximum capacity, just as the man with the thin seam does.

At the Cincinnati convention (May, 1931) we will see cars 15 ft. over-all, 7 ft. wide and 38 in. off the rail with a water level capacity of 200 cu. ft. There are a great many cars of this height or greater, with less than one-half that capacity.

In determining the number of cars needed, the turnover is not very important. As Mr. McElroy says, a mine may have a good car turnover and be inefficient in every other department on account of a lack of cars; and, as he also states, cars are the cheapest form of storage. This is true, especially where breakage of coal is a factor. When car turnover begins to go above normal for a given mine, it is ripe for a series of time studies to see whether production is being lost on account of not having enough cars.

In the coal-mining industry we are accused of being notably behind most other industries in modern methods of cost cutting by mass production. A few years back Eugene McAuliffe gave a paper at the Cincinnati Coal Show in which he stated that there was no reason why a coal mine should not work continuously, as a steel mill does, or a power plant, a blast furnace, etc.; that expensive tipples, washeries, locomotives, etc., should work 24 hr. instead of 8. Most of the comments heard in regard to the paper were to the effect that it was extremely visionary and that any such scheme could not be applied to the average coal mine. There are mines in West Virginia today working three loaders in a 14-ft. place, in 5 ft. of coal, taking three cuts per 8-hr. shift and the place is working continuously, or three 8-hr. shifts.

One mine loads 2000 tons from 12 working places in 24 hr. They find that they have about three times as many locomotives and cutting machines as they need. At this particular mine they should not have to buy any rail or trolley wire for the next 10 years. The dispatchers have already been cut off, as they use only one main-line motor per shift and it goes to one sidetrack. They have enough mine car storage for the 2000 tons, and the tipple crew dumps this coal in 4 hr., after which these men may be used for other work.

The whole scheme seems practicable now instead of visionary, and the men appear to like it. I am merely pointing out this turn that the mining industry is taking in one coal field. If it is successful we will still want big cars, and more of them than we needed before, but some of the points brought out in Mr. McElroy's paper will not fit into the scheme.

The paper shows that a great deal of work has been done in the field and much time and thought given to assembling the data, but the thought that occurs to me is that while the time studies and data were collected in a representative group of West Virginia mines, it is this representative or average mine's methods that we are trying to get away from.

Dry Cleaning of Coal in England

By Kenelm C. Appleyard,* Birtley, England

(Pittsburgh Meeting, September, 1930)

In introducing to an American audience a description of the work done in dry coal separation in England and in Europe generally, it is perhaps desirable to give a short history of the development outside the United States of the particular processes with which the author is connected, and perhaps to stress a little the differences in the nature of the conditions to be faced.

England actually presents one set of problems and the Continent of Europe another, since the character of the coals to be treated is entirely dissimilar, and the attitude of the coal industries towards coal preparation is also dissimilar. England for generations has been working by hand good seams of clean coal of the highest possible quality and has not felt any pressing need to clean its coal. Most Continental countries, however, especially in Central and Western Europe, have been working friable and often very dirty coals for so long that a washery is accepted as a part of the normal equipment of a colliery, with the result that the art of wet washing has reached a high state of perfection, particularly in Belgium and Germany.

The advent of machine mining has intensified the problems to be faced in all countries, and in England has been coupled with the working out of the best seams in the older coal fields, thus bringing about the necessity of working thinner and dirtier seams with resultant difficulties as regards marketing the products, while Poland, with clean and well prepared coal, has in recent years been attacking established British markets in Scandinavia.

In recent years, therefore, interest in coal preparation has been steadily gaining ground in England, although it has not yet reached Continental dimensions. Broadly speaking, the differences in preparation problems between England and the Continent boil themselves down to one of size. In England, for example, with the exception of the Kent coal field, the proportion of coal of a size less than ⅛ in. to the coal less than 2 in. is very much less than on the Continent, and since the principal difficulties in dry cleaning lie in the fines it is easier to produce good figures on

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* Managing Director, Birtley Iron Co.

English coals and to build compact plants for large outputs. On the other hand, Continental coals are not usually so dirty in the fines as many English coals, so that to some extent the difficulty is ameliorated, and while the plants are perhaps more complicated, adequate results can be obtained by the dry process.

The gas coals of the County of Durham are famous for their high qualities, and it was therefore in this field that pneumatic separation started in England towards the end of 1925. The machines used were American machines of the S. J. type, and there is no need therefore to describe them in detail.

It may be said, however, that observation of the earliest separation plants in America had led the author to the conclusion that, generally speaking, they were not at that time sufficiently stable and that the separation suffered from building vibration. As a consequence, no European plants have been built which are not housed in buildings of steel construction with brick-filled walls and reenforced concrete floors, all the steelwork being of heavy character and strongly braced.

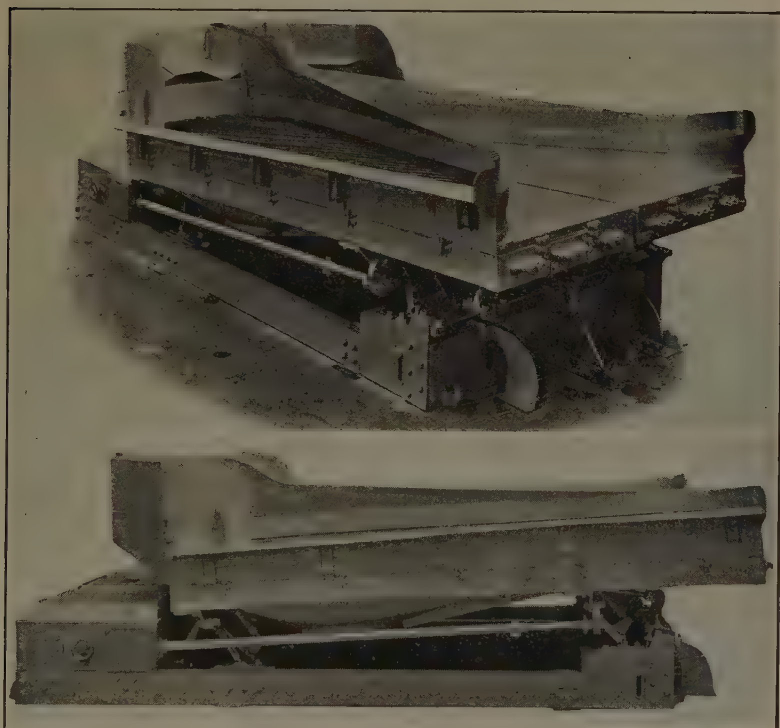
During 1927 the American Y-table succeeded the S. J., and from this point developments in the machines ceased to run parellel in the two countries. A certain amount of restriction on the surface of the Y-table caused capacity difficulties with some types of English coals carrying heavy inherent moisture and containing large quantities of sluggish shale. Therefore, experiments were carried out with decks utilizing straight-line spillage, and finally the present type of V-deck was evolved, having a curved banking bar with straight deck edges.

Coincidental with the work on the standard machine, a supermachine of the same type was developed with very high capacity up to 1½ to 2 tons per square foot of deck area. Mechanical troubles, due to the reciprocation of heavy loads, were experienced in the early stages of this machine, but these have been overcome to the extent that a single unit, capable of treating 100 tons of coal per hour of normal English grading proportions, runs smoothly at 400 r.p.m., with driving and eccentric shafts of only 1¾ in. diameter.

Figs. 1 and 2 show this machine, which is about to be superseded by a more refined and simplified article in which the airbox itself acts as a rigid girder and extends over the whole surface of the machine, while the half decks are adjustable for side inclination within the box. The old-type double cone drive is abolished and replaced by a single shaft drive with split-pulley type of variable-speed regulator on it. The whole machine is perfectly balanced by a torsional spring device, and, in accordance with the practice adopted from the first in England, no plain bearings are used on the machine. It is essential that machinery of this type should be well constructed and contain the highest class of workmanship and materials.

No Y-tables are now at work in England; they have all been converted to the V-type, with increases in capacity up to 30 per cent. Some 35 cleaning plants of this type are now at work in Europe, with capacities running up to 250 tons per hour, while approximately 40 plants are at work in countries outside Europe, these 70 or more plants containing 221 machines.

It was soon found that the possession of an efficient cleaning machine was only the beginning of the business, and that before it would be



FIGS. 1 AND 2.—TWO VIEWS OF V-DECK TABLE WITH CURVED BANKING BAR AND STRAIGHT DECK EDGES.

possible to reach the ideal plant—clean, foolproof, elastic and efficient—many problems had to be solved, two or three solutions being required in some cases for different types of coal. Certain of these difficulties were obvious at the start, but others cropped up only when troubles arose in operating commercial plants.

The obvious special problems and items of subsidiary plant which required development and perfection in 1925 were:

1. Aspirators.
2. Dust collectors and dust disposal.
3. Screens and screening.
4. Vibration of buildings.
5. Increases in capacity of units.

Since then, the following problems have been shown up as having a considerable effect on separation efficiency, and the final satisfaction of the customer who is to receive the fuel as well as the colliery company installing the plant:

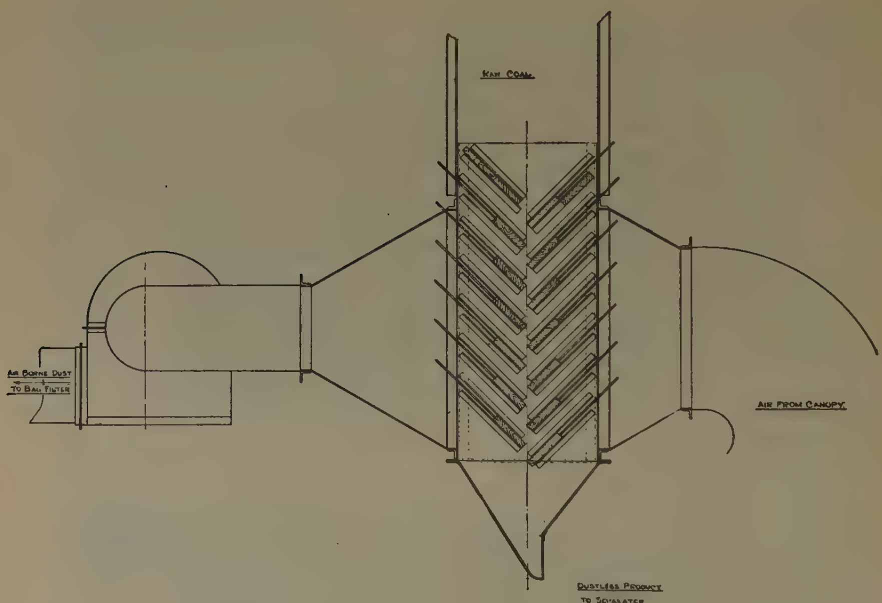


FIG. 3.—ARRANGEMENT OF EARLIEST TYPE OF ASPIRATOR.

6. Bunkering of raw, sized and clean products.
7. Breakage.
8. Size segregation before cleaning.
9. Blending.
10. Variation in mixtures of raw coal from different seams.
11. Distribution of dirt, i.e., whether bulk is in larger sizes or in fines.
12. Plants for small users.
13. Difficulties of plants where numerous accurately sized qualities are required.
14. Reduction of air consumption.

Each of the above problems has involved research and experiment over some years, and while large strides have been taken, investigation still continues, the main problems of study being as follows:

Reduction in power consumption, particularly with regard to dust collection.

General simplification of plant.

Treatment of unsized coals with the object of getting equal efficiency throughout the range.

Perfect treatment of fines.

Treatment of coals with wide variations as to dirt and moisture content.

Elimination of breakage.

Conveyance of dust.

Disposal of dust with very high ash contents.

A survey of the experimental work must necessarily be short if this paper is to be kept within reasonable dimensions, and it is therefore proposed to indicate the general lines of the original apparatus, sketch briefly the intervening steps without detail, and describe more fully in each case the practice obtaining at present.

Aspiration

The necessity of dedusting coals before cleaning, whether by wet or dry process, is now fully recognized, and its importance when treating dry fines can hardly be overestimated.

The earliest type of aspirator used by the author in England is shown in Fig. 3. It consists of a succession of cascades designed to set up the dust contained in the coal stream so that it may be carried away by a strong current of air. An example of the results obtained is given in Table 1, which shows the grading of the coal fed to an aspirator in ordinary commercial use, together with the grading of the discharge and of the dust collected. The coal is dry and in this case the raw-coal ash content averaged 12.62 per cent. for all sizes, while the ash content of the aspirator dust was 8.32 per cent. It is clear, therefore, that a certain amount of heavy material is rejected by the aspirator and fed to the cleaning table, where it is removed by the ordinary process of pneumatic separation.

The defects of this aspirator were that, owing to small clearances, it was liable to choke up when pressed hard, and that with damp coal the cascades built up, thus reducing the air openings, increasing the velocity of air passing through, and so removing too much in the way of large particles and too little dust.

It was succeeded by the aspirator shown in Fig. 4, which is also of the cascade type but with steps in one direction only. Table 2 gives the grading schedule. This was a distinct improvement, but in this one also damp coal would build on the cascades and alter the air velocities. Scrapers on the steps worked by hand modified the difficulty, but regula-

tion was difficult and after much work aspirator No. 3, representing present practice, was produced. This is shown in Fig. 5, with grading figures in Table 3.

The raw coal to be aspirated is retained in bunker *A* by means of an adjustable door *B*, of which the height is adjusted so that the raw coal rests on the adjustable plate *C*, which forms the lower side of a horizontal slot open to atmosphere.

When the suction fan *G* is started, the pressure within the aspirator is reduced below atmospheric pressure, and the difference is sufficient to alter the angle of rest of the coal on plate *C* and cause it to feed across the vertical flue of the aspirator. Secondary air, sufficient to give the required results, is then admitted below the raw-coal feed through adjustable ports *D*. The dust separated out of the coal passes into a baffled settling chamber *F*, where the bulk of it is deposited; only the finest particles passing through the fan *G* to the bag filter, or, in cases where this practice is desirable, through the filter to the fan. It is, of course, necessary to discharge the dust-free coal and the settling-chamber product through air seals, and it is found in practice that weighted flopper valves (*E* in diagram) function simply and efficiently.

The feed to the aspirator is automatic, the coal ceasing to flow over plate *C* as soon as the fan is shut off; the air seal at the feed, therefore, is maintained automatically.

Table 1.—*Grading Figures of Coal Using Aspirator No. 1*

Size of Coal, In.	Percentage Weight		
	Raw Coal	Aspirated Coal	Dust
+ ⅛	10.7	16.1	0.5
⅛–1/16	30.4	41.5	2.6
1/16–1/32	23.3	21.3	15.8
1/32–1/64	17.3	13.7	25.3
1/64–0	18.3	7.4	55.8
	100.0	100.0	100.0

Table 2.—*Grading Figures of Coal Using Aspirator No. 2*

Size of Coal, In.	Percentage Weight		
	Raw Coal	Aspirated Coal	Dust
+ ⅛	38.7	48.6	0.6
⅛–1/16	20.5	27.2	6.0
1/16–1/32	14.2	14.3	21.0
1/32–1/64	9.2	8.1	32.8
1/64–0	17.4	1.8	39.6
	100.0	100.0	100.0

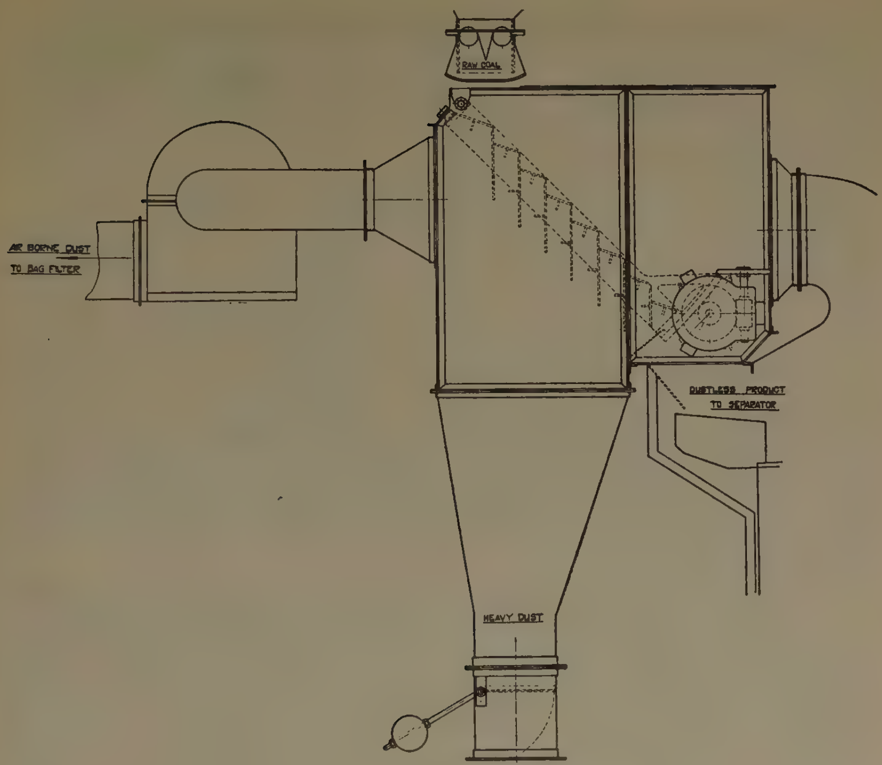


FIG. 4.—ARRANGEMENT OF ASPIRATOR No. 2.

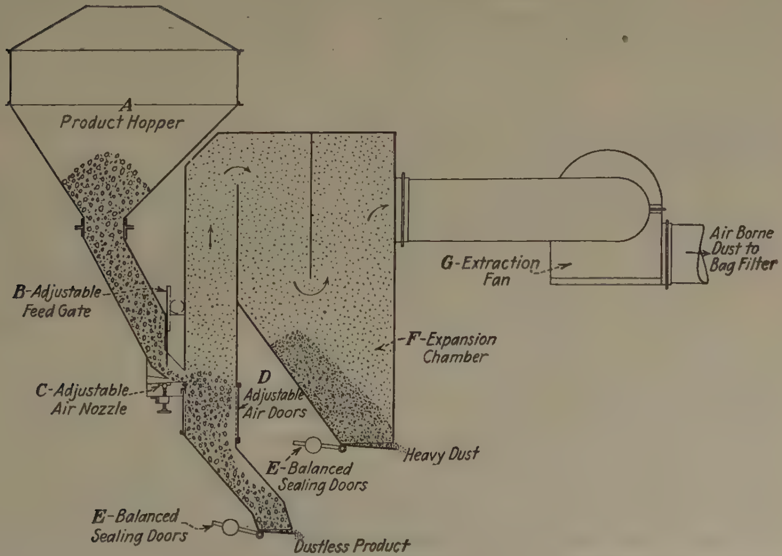


FIG. 5.—ARRANGEMENT OF ASPIRATOR IN USE AT PRESENT.

TABLE 3.—*Grading Figures of Coal Using Aspirator No. 3*

Size of Coal, In.	Percentage Weight		
	Raw Coal	Aspirated Coal	Dust
+ 1/8	38.7	46.2	
1/8–1/16	20.5	23.2	0.8
1/16–1/32	14.2	17.5	4.0
1/32–1/64	9.2	10.2	16.8
1/64–0	17.4	2.9	78.4
	100.0	100.0	100.0

Normally, in applying this apparatus to an existing washery, about 8 ft. of head room will be required. For normal dedusting practice, less than 200 cu. ft. of air per minute at 3-in. water gage is required for each ton per hour of raw feed, and 1 in. width of feeding slot should be allowed for each ton per hour.

The first aspirator of this type was put to work recently at Dahlbusch in Germany. The fines below 10 mm. are screened out before the coal goes to the Rheolaveur washery, and pass through a primary louvre-type aspirator with a view to removing the dust below 0.5 mm. before sending the fine coal to be cleaned on Birtley pneumatic separators. When this dry-cleaning plant was installed, it included a cascade-type aspirator to remove the dust that escaped the first aspirator, but intermittent concentrations of both moisture and dust were encountered, with which neither aspirator could cope, and as a result high-ash dust found its way into the cleaned coal periodically. The replacement of the cascade aspirator by the new type has proved successful in dealing with the peak loads, so much so that the only indication of high moisture conditions on the separators is the darker color of the damp shale.

A series of samples showed that the raw coal below 10 mm. contained 20 per cent. of dust below 0.5 mm.; the coal after the first-stage aspirator contained 13.8 per cent. of dust, which was reduced to 2.3 per cent. in the coal leaving the second-stage aspirator. Fig. 6 shows how the various sizes in a sample of coal are distributed between the aspirated coal and the collected dust after a single treatment to remove the dust below 1/64 inch.

The curve shows the essential difference between aspiration and screening. In the present case a high percentage of the dust has been removed at the expense of a certain amount of oversize, whereas if a 1/64-in. screen could have been used it would have kept the oversize in the aspirated coal at the expense of an inferior efficiency in dust removal. This removal of oversize particles has distinct compensations,

as a study of the ash contents of the various size fractions of the raw coal, aspirated coal and dust shows.

Table 4 gives analysis of the three products obtained from a test on a Northumberland steam coal.

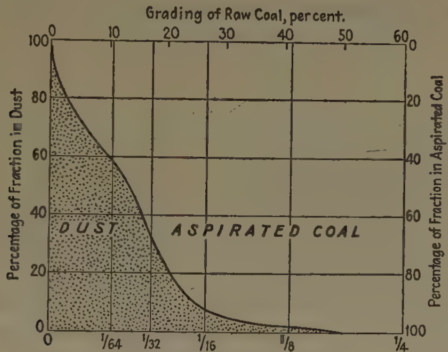


FIG. 6.—DISTRIBUTION OF SIZES IN A SAMPLE OF COAL.

TABLE 4.—*Ash Contents, Northumberland Steam Coal*

Size of Fraction, In.	Ash in Raw Coal, Per Cent.	Ash in Aspirated Coal, Per Cent.	Ash in Dust, Per Cent.
$+\frac{1}{8}$	14.80	15.55	12.50
$\frac{1}{8}$–$\frac{1}{16}$	14.95	17.20	9.45
$\frac{1}{16}$–$\frac{1}{32}$	17.00	23.40	11.20
$\frac{1}{32}$–$\frac{1}{64}$	19.40	31.00	17.00
$\frac{1}{64}$–0	24.95	33.65	22.10

This aspirator functions satisfactorily for commercial purposes, and is not nearly so susceptible to change of efficiency with variations in moisture content as the earlier type.

The power required for an aspirator passing 20 tons per hour of $-\frac{1}{8}$-in. coal is approximately 3.5 horsepower.

DUST COLLECTION AND DISPOSAL

A glance at the early pneumatic plants, with their cyclones belching forth dust, was enough to indicate the urgent necessity of tackling the problem of getting 100 per cent. dust collection, concentration and disposal.

The first English plants were equipped with large bag filters into the top box of which the dust-laden air was blown. They were economical as regards power consumption and efficient in operation, but required

a great deal of space and height, while sharp particles, when more or less fired into the tops of the tubes, caused a good deal of trouble by holing the fabric rapidly, thus entailing a high upkeep cost.

The next step was to add a cyclone to the range to remove the heavy particles before passing the dust-laden air to the bag filter. This also was successful, but entailed heavy capital cost and high power consumption.

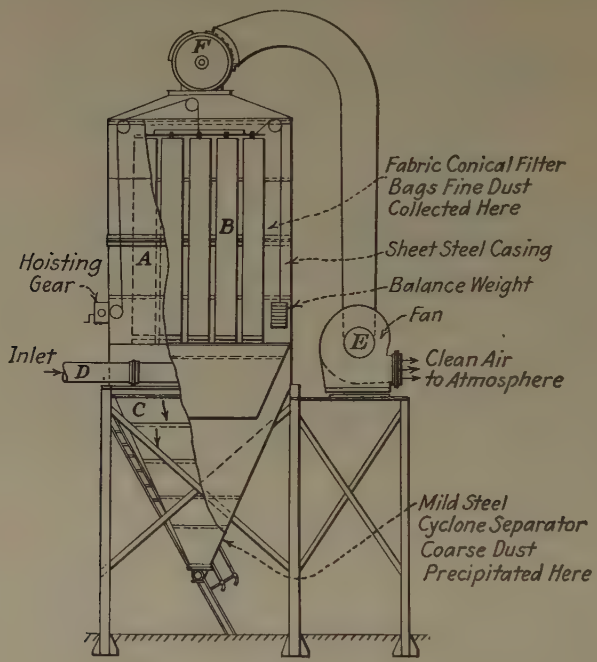


FIG. 7.—WARING DUST COLLECTOR.

A very full survey was then made of the following methods of dust collection and concentration:

Steam saturation,
Water settling,
Perforated rotary cylinders,
Circulation through grit-carrying bands,
Electrostatic,
Streamline filters,
Oiled copper-tube type filters,
High-pressure fabric filters with shaking gear,
Fabric filter with reversal of air currents,
Low-pressure fabric filter with shaking gear.

In almost every case some cause was found, often high capital cost, for abandoning the method.

Eventually use was made of the Waring filter with pressure fan, and finally with suction fan, the latter being the method in standard use

at present. Properly designed, the operation of this filter is practically perfect. No dust passes through the fan; the cleaning plant is sufficiently free from dust to make it possible for anyone wearing a light suit to enter the building without fear or a dustcoat, and the whole of the dust is concentrated in deaerated form at single points for disposal.

Fig. 7 shows that a vacuum is created inside the case *A* above the cone line, and, by suction through the filter tubes *B*, the cones *C* and trunking *D* right back to the table hoods are under vacuum conditions. The dust-laden air is therefore drawn into the filter at the level of the cone tops, the heavier particles are discharged by centrifugal action and expansion as in a cyclone, while the fine dust particles pass up inside the filter tubes, where they are deposited, clean air only passing away through the fan *E*.

If the dust deposit becomes sufficiently heavy inside the tubes to increase the resistance to an undue extent, the rotary valve *F* is operated either automatically or by hand, and the casing of the filter is opened to atmosphere. The filter tubes collapse instantly and shake off their load of dust, thus restoring the normal resistance.

The filter discharge is normally of the balanced-valve type, and dust is easily delivered to conveyors or suction systems for mixing back into cargoes or burning.

The performance of the Waring filter is as follows: 50,600 cu. ft. per min., 2.25 in. water gage in cyclone, 4.0 in. water gage in case, 48 horsepower.

Screens and Screening

The first screens used for fine work on air plants in England were the old Mitchell-type vibrating screens; some of these are still at work, but are not efficient, particularly under heavy loads, and where prescreening is adopted the Hum-mer screen is now the standard. The initial cost of installation, however, is high, particularly in Continental installations treating large quantities of fines, and where conditions are suitable every effort is made to adopt the re-treatment method whereby unsized coal is fed to a first machine and screening takes place between cleaning operations. This general method will be discussed later, but it can be said here that the Arms screen has been used successfully for this type of work where the smallest screen opening is not smaller than, say, 3/16 in., and the coal is quite dry.

Vibration of Buildings

An indication has already been given as to how this problem has been handled so far as the buildings themselves are concerned. Admittedly, the capital cost is higher than when using timber construction and corrugated sheet, but in European countries standards of this kind are

universally accepted and are well worth the money. In addition, however, there has always been the problem of balancing the machinery to avoid transmitting vibration to buildings. So far as the air tables themselves are concerned, this is done by means of coiled springs and a torsion balancing arrangement, which can be seen underneath the machine in Figs. 1 and 2. All parts, such as pulleys, are accurately balanced during manufacture, and there is finally left the screen problem and others such as elevator vibration.

With Hum-mer screens properly installed in a solid building, the vibration is not such as to affect separation or cause building movement. Shaking screens of the Arms and other types, however, are not permitted to be installed except in the same line as the air tables, since separation is undoubtedly affected where such screens operate at an angle with the direction of table jig.

Elevators are universally of the continuous bucket type, with square or hexagonal tumblers. Trouble has occasionally been caused by placing an elevator at right angles to the air tables, owing to the jar produced as the links go over the tumblers, particularly with square tumblers. This is difficult to modify, and belt conveyor feeds are now always used where circumstances permit, since these give a smooth uninterrupted feed to a plant.

Increases in Machine Capacities

This is entirely a question of machine design and the correct use of the principles involved in the art of specific gravity separation. It is interesting to note, however, as an indication of the advance in knowledge during the period, that the increased capacity per square foot of deck area between 1925 and 1930 is approximately 35 per cent., while keeping the same high standard of efficiency, while the range of sizes treated without decreased efficiency has been increased from 2:1 to 4:1.

Simplification of Operation

It has been the practice in England to use unit drives wherever possible, and since the ideal to be attained is that the plant operator shall have no other steady duty than to keep an eye on his machines, it became necessary to make all electrical equipment as nearly foolproof as possible, and automatic in operation. As a result, all European plants are equipped with automatic switchgear and push-button control, motors being interlocked so that the failure of one, driving perhaps a conveyor, will instantly trip out all others affecting it back to the nearest bunker; that is to say, a clean-coal conveyor motor on tripping out will instantly stop all table motors but not the screen and raw-coal elevator motors, since these merely feed the bunker storage above the tables.

On the other hand, if the sized-coal bunkers referred to filled up, overloaded a screen and tripped out a screen motor generator, then the elevators, conveyors and everything back to the source of raw-coal supply would be automatically stopped.

These automatic equipments are fitted with thermal overloads for each motor with illuminated telltales which light up as each motor starts up and black out on failure.

There is one main start button and one main stop with emergency main stops in selected places in the building, while each motor has its own stop button on the board. When the main start button is pressed a klaxon horn is sounded for ½ min. followed by 30 sec. pause to allow any man working on machinery to receive warning and stand clear.

Figs. 8 and 9 show the type of apparatus used.

Bunkers

In England it is sometimes difficult to persuade colliery owners that large bunker accommodation is a profitable investment. This is due, perhaps, to the financial position of the coal industry rather than to conservatism, but the attitude is in marked contrast to the Continental one, where large bunkers are everywhere demanded, it being by no means unusual to find 3000 or 4000 tons of bunkerage for raw and washed coals in washeries of which the capacity is only 150 tons per hour.

Certain air-table plants in England have been built without adequate raw-coal bunkerage between the source of supply (pit or railway cars) and the preparation plant, with the result that unsatisfactory operation has ensued, particularly owing to pit fluctuations.

It is now standard practice, when taking coals direct from a pit, to install a raw-coal bunker with a minimum capacity of one hour's feed to the plant (100 tons for a plant treating 100 tons per hour). Where the feed is from railway cars, the point is not so material.

With plants arranged for prescreening, sized-coal bunkers should never be of smaller capacity than 20 min. feed, and even this is insufficient where a mixture comes from seams that vary considerably in grading characteristics.

Everything that assists in giving a regular feed to a plant and a regular feed of sized coals to the machines is to the good, and a paying investment, since experience has proved that when the flow of material to the machines is interrupted the interruption takes place at the expense of efficiency.

Blending

Closely associated with the bunker question is that of blending.

The ideal raw material to feed to a cleaning plant, whether wet or dry, is one in which the coal characteristics, both as to grading and

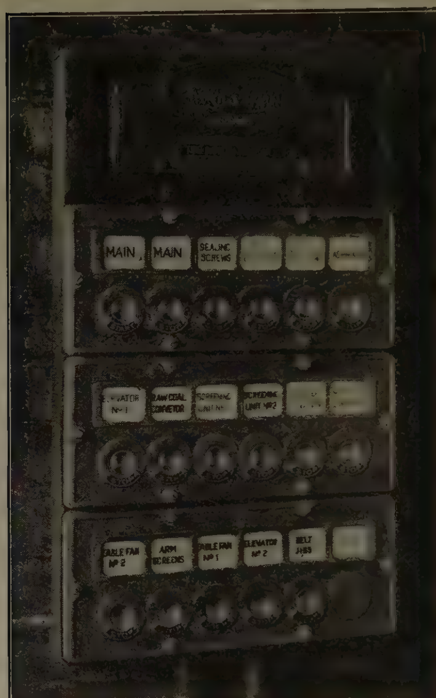


FIG. 8.—PUSH-BUTTON CONTROL BOARD.

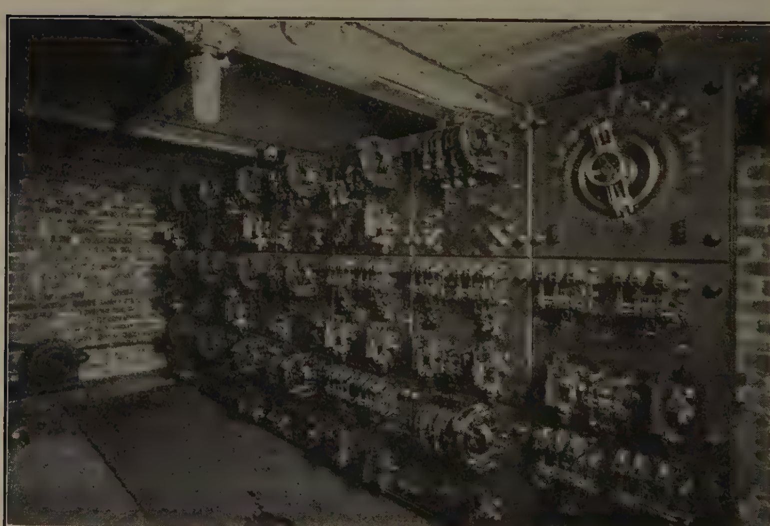


FIG. 9.—AUTOMATIC SWITCHGEAR EQUIPMENT.

refuse content, do not vary. Exactitude in this matter is not, of course, possible in practice, since coal from a single seam varies, but where it is desired to clean a mixture of two or more seams the tendency in Europe is to have a raw-coal bunker for each seam and mix them in regular proportions, thus not only improving the efficiency and operation of the cleaning plant but also making absolutely standard fuel.

This method has recently been adopted at a plant cleaning 100 tons of coal per hour by air tables, the cleaned coal being regular in quality and passing to the main 3000-ton storage bunker for a large battery of coke ovens. In this case the coal is brought in by car from several collieries, each quality being dumped into its own underground bunkers and the required proportions from each bunker fed by jigging feeds on to mixing conveyors which deliver to the main elevator of the pneumatic plant.

Practically all the normal fluctuations are done away with in this manner and the resulting fuel is cleaned to within approximately 1 per cent. of the fixed ash without difficulty.

Size Segregation before Cleaning

Allied also to the question of bunkers is the subject of size segregation before cleaning, and so far as the author is aware this subject has not generally received the attention that it merits.

If raw coal is delivered into a bunker from a direction at right angles to the direction of feed *from* the bunker, it will be found invariably that in a sample taken across the full width of the feed from the bunker there is a very substantial difference in proportion of sizes between one side of the sample and the other, due to the natural size separation within the bunker from the original feed.

Endeavors to cure this by means of spirals within the bunker failed owing to the separating action of the spirals, and wherever possible plants are now designed so that right-angle feeds are avoided; where this is impossible bunkers are so shaped at the tops as to baffle the throw from the original belt or elevator and to deliver the whole mixture over the opening at the bunker bottoms.

For some time it has been the practice in England to design bunkers with unsymmetrical sides in order to prevent arching; this, however, has now been abandoned for the reason that segregation always takes place.

The following table shows the grading of coals fed to the two sides of a single air table, the difference being due solely to segregation:

	1–¼ In.	¼–1/16 In.	1/16–0 In.
Right hand	48	28	24
Left hand	22	26	52

The interference in efficiency with differences of this kind may be imagined and the resulting troubles were easily cured by baffling within the bunker to remove the segregation.

Breakage

With gas and coking coals the question of degradation is not regarded as being of any great importance in Europe. With steam and similar qualities, however, it is another matter. It is probable that the development of cleaning-plant design suffered somewhat because all the early installations were for gas coal and the breakage question was ignored. The problem is a most difficult one. Cleaning plants require height in order to get the advantage of gravity flow, and with soft fuels height involves breakage.

The points at which degradation occurs most are (1) at the elevators, (2) in bunkers, (3) in the chutes from the air tables to the gathering conveyors of cars. It has already been said that belt conveyors are installed in preference to elevators. Where the latter are unavoidable they are fed mechanically, and at the discharge end the chute is arranged to catch the coal at the point in its trajectory where the least drop is obtained, while generally the elevator delivery chute is cascaded so that a certain amount of coal is retained in it as a cushion. With regard to the second point, spirals have been discarded as ineffective, except for coals which have already been cleaned, and double vertical cascades with open backs are used. These undoubtedly ameliorate the breakage difficulty, but are not perfect by any means. Nothing, of course, can improve degradation caused by coal moving on itself in a bunker under service. So far as point 3 is concerned, most air tables show a serious drop for the fuel from the delivery edges of the decks into the gathering chutes leading to disposal points, and investigations have shown that breakage is most serious here.

A great deal has been done in English plants by cascading the hoppered fenders between deck edges and chutes proper, but the tendency now is to build semicircular chutes on to the air tables so that clean coal instead of dropping from the deck immediately is carried forward to a chute immediately adjoining the middlings chute and taken thence at a slow angle to the disposal point, thus the gathering together of the clean coal is actually done on the machines instead of by dropping a considerable distance into a stationary hopper.

The real solution of the problem of actually getting into cars carefully sized coals is, where circumstances allow, to have a final screening operation, the sized coal going direct from screens to cars.

The general problem of whether presizing or postsizing is the better will be discussed later, but when carefully graded coals are required for a market there is a strong argument for postsizing.

Cleaning

In considering the design of a cleaning plant it is not possible to start with a specific general design which will be applicable to any class of coal. Indeed, the question of design should not be approached until the fullest data are available as to the coal to be treated. With this in mind, the normal procedure of the companies with which the author is associated is first to examine in the laboratory a small sample of, say, 200 lb. in order to get a general idea of the coal characteristics. This is followed by a carefully run test on commercial machines, regular and accurate samples being taken and analyzed. The quantity used for this test is as a rule 45,000 lb. This is followed by a study at the colliery of the coal as it is drawn, to note what fluctuations in seam mixtures, gradings and refuse content take place from hour to hour and from 15 min. to 15 min. Any apparent discrepancies between the three sets of figures are checked up and it is then possible to determine how the coal is to be treated and what guarantees can be given, all plants being installed on guarantees.

It may be said that all this is much more elaborate than is required in connection with wet-washing plants, but experience has proved that an accurate knowledge of the facts is by far the best thing to start on, and it is certainly necessary in order to enable cleaning plants to be satisfactorily planned. It is unnecessary to discuss the possibilities and limitations put upon the designer by circumstances of site, transport arrangements and so on. These are common to all types of industrial plants and must be solved for each particular case: what has to be decided is the amount and type of cleaning machinery required, how it can be adapted to suit the circumstances indicated above, and what is the best method of putting on to transport the grades and qualities of coal that the colliery owner desires to sell to his customers. It is necessary, therefore, to know what the colliery sales department requires in these directions, and with this and the test information available the first matter to be decided is whether the coal shall be sized before cleaning, during the cleaning process, or after it.

The problem has to be considered from several angles:

1. If a very clean coal is required right through the size ranges down to the dead fines, it is usually the practice to presize it into grades approximately as follows: 2 to ½ in.; ½ to ⅛ in.; ⅛ to 0 in. These are rarely adhered to exactly, because the screens are changed shortly after a commercial plant begins to operate, in order to balance the feeds going to each table and give them the proper loads, but with normal coal the ranges of 4:1 give excellent cleaning. If the −⅛-in. material is very dirty, it may be necessary to allow for smaller tonnages to be fed to the fines machines and to increase the usual number.

2. If only a reasonably good coal is required for market purposes, say a coal with an ash content about 3 or 4 per cent. above the 1.5 sp. gr. float ash (and this is very often the case with steam coals), it is frequently the practice to feed the unsized coal below 2 or 2½ in. to a single machine, pass the cleaned product to a screen with perhaps a ⅜ or ¼-in. mesh, take the oversize into cars or bunkers, and retreat the undersize; a further sizing and cleaning operation would actually give a coal clean enough for the requirements of paragraph 1, and therefore for condition 1 a decision between the two methods is usually governed by tonnage considerations.

As an example, if the raw coal to be cleaned is such as to give a very light load to certain machines and very heavy loads to others when prescreened into the normal ratios already indicated, it may be possible to get round this difficulty by adopting the re-treatment method described for condition 2.

If neither of these methods suits the coal and the tonnages, it may be desirable to make a first split at, say, ⅝ in. and treat the +⅝-in. and −⅝-in. by re-treatment methods.

It is impossible to generalize; each problem must be decided on its own merits.

3. With certain classes of coal it is possible to obtain the desired results by treatment on one machine without any sizing. This is "rough cleaning" and removes as a rule all the visible stone while doing a certain amount of good work on the smaller sizes. Where a raw coal is cleaner in the very finest sizes, a single-stage treatment below 1 in. gives excellent results.

Where gas and coking coals are concerned, the cleaned grades are often remixed, and the question of making carefully sized qualities does not arise: in such cases the coals are merely gathered together from the machines on to a conveyor and delivered to cars.

Sometimes, however, the normal table sizes are also the market sizes for certain classes of coal and the usual degradation from bunkers, tables and conveyors causes troubles when the coals are not finally screened before being loaded into cars.

In the writer's opinion, when accurately sized products are required for the market, the coals should first be sized to suit the cleaning process (if this is necessary) and finally sized after cleaning and immediately before loading. In this way any degradation due to processing is picked up and the designer and operator have a free hand to arrange the tables in the way that best suits the circumstances.

"Middlings" should always be recirculated. Whether the coal has a high percentage of intergrown and high-ash material or not, the dividing line between stone and clean coal must be cut out if good results are to be obtained.

The results shown in Table 5, taken from a Polish plant, at which inexperienced Polish operators were experimenting in order to discover the best operating conditions, illustrate this point clearly.

TABLE 5.—*Results of Cleaning at Polish Plant*

NO RECIRCULATION OF MIDDLINGS

	Pea Per Cent.		Size II—Nut Per Cent.	
Raw-coal ash	7.4		7.00	
Yield: coal	93	containing 4.9 stone	89.8	containing 3.25 stone
middlings	4.5	containing 25.0 stone	5.5	containing 24.50 stone
refuse	2.5	containing 26.0 coal	4.7	containing 21.00 coal

WITH MIDDLINGS RECIRCULATION

	Pea Per Cent.		Size II—Nut Per Cent.	
Raw-coal ash	6.33		5.71	
Yield: coal	95	containing 2.02 stone	95.75	containing 1.03 stone
refuse	5	containing 1.62 coal	4.25	containing 1.00 coal

MACHINE DESIGN

This is not the occasion to discuss fully the details of machine design, but a few points of interest are worth mentioning.

Many efforts have been made to develop air tables to operate without the banking bars against which the shale is thrust by the motion of the machine; all our research in England indicates that if capacity and clean stone are required, it is necessary to have this banking bar, and the increases in capacity from year to year already referred to have been obtained by careful research into the form which the banking bar should take—whether angular or curved, and the exact curves required—coupled with a careful study of deck motion and shape, together with methods for regulating the depth of coal bed on the machine.

The present form illustrated in Figs. 1 and 2 is the outcome of these years of research, and although these machines are now undergoing simplification mechanically the general design of the deck itself is maintained as the best available today in Europe. Accuracy of workmanship and excellence of materials are *sine qua non.*

Lost motions or irregularities of action are felt immediately in lost efficiency of the machine, and one of the principal difficulties in its detailed design has been to ensure that joints are made and remain tight. The elimination of joints is one of the reasons for the remodeling which is at present being carried out. Every moving part and joint is accurately machined in order to avoid these troubles.

Much work has been done in the direction of balancing machines in themselves to avoid transmitting vibration. The reciprocating weight has been steadily reduced until in the latest machine, with 80 to 100

tons per hour capacity (with 45 sq. ft. deck area) it will be only 2200 lb. It is so balanced as to require a light driving shaft of only 2⅜ in. dia., and although it is impossible to exactly balance a machine of this

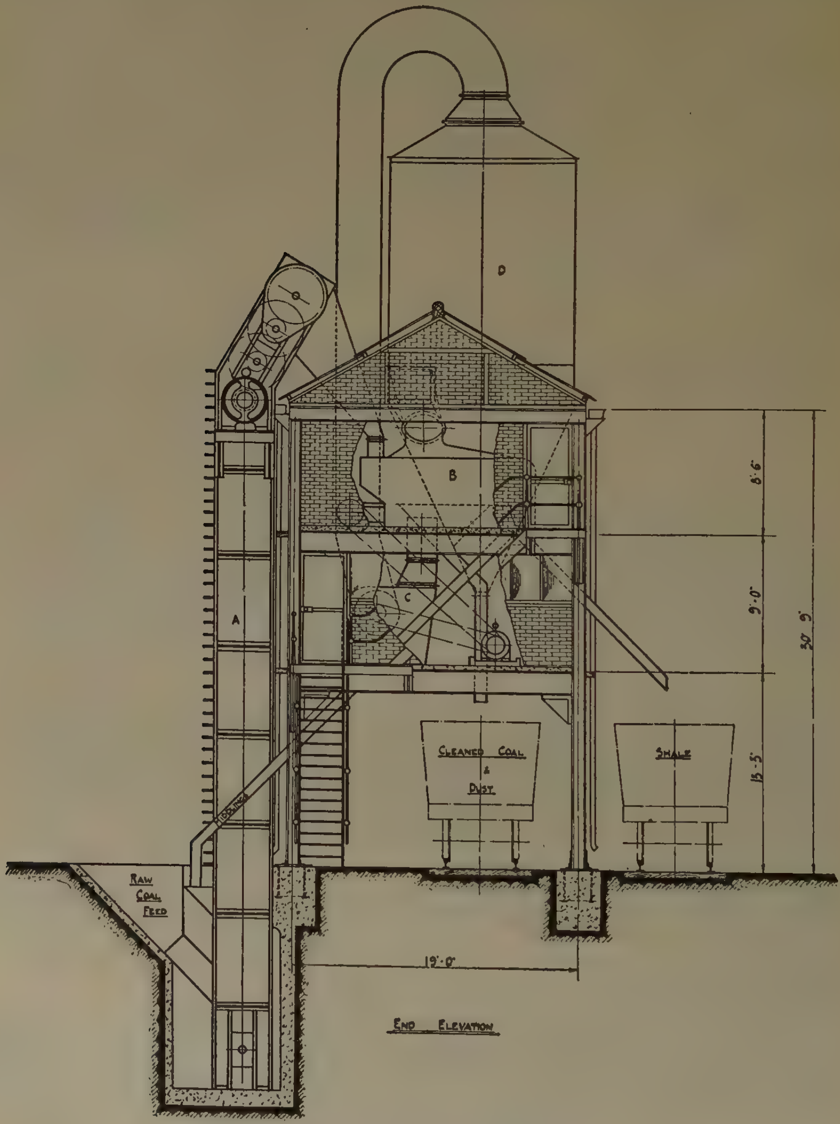


Fig. 10.—Standard pneumatic separating plant.

kind, which has a fluctuating load, all this work, plus very stiff buildings, produces vibrationless cleaning plants, despite the fact that the machines are working on floors that are seldom less than 40 ft. above ground level and are mounted on 4-in. concrete floors.

On the table illustrated in Fig. 1, it is possible to clean two sizes of coal on one machine and several plants of this type are now under construction in England, thus meeting to some extent the problem of the small colliery that desires to clean a fairly wide range of coals to a low ash figure. For the same type of colliery where rough cleaning is required,

FIG. 11.—RE-TREATMENT TYPE PREPARATION PLANT.

there has been developed in England the so-called "Standard Plant"—"so-called" because although no two are alike, the general dimensions of buildings, materials and machines remain the same and are therefore cheaper to design and manufacture. Fig. 10 shows the outline of one

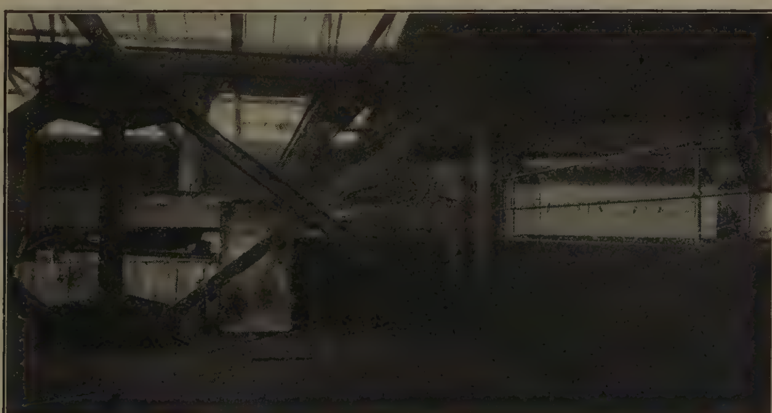


FIG. 12.—SECTION OF INTERIOR OF PLANT SHOWN IN FIG. 11.

of these plants and shows that one fan looks after the pressure air required by the table and also the suction required by the dust-collecting plant.

A further tendency very noticeable in Europe is that of designing the whole preparation plant—both hand-picking and dry cleaning—in one unit, and Figs. 11 and 12 show the first of the kind to be built in England.

In this case, all operations are carried out under one roof and the dry-cleaned smalls below 2 in. are eventually fed into a central division of the picking belts, thus ensuring a proper mixture of sizes for the gas-coal cargoes shipped by this colliery. In this case the re-treatment method was also used for the first time with complete success.

Operation of Commercial Plants

It is particularly difficult to produce sets of figures for a number of cleaning plants which will be in any way comparable, since the conditions under which each colliery runs its affairs differ considerably, methods of charging expenses vary, and some collieries clean the whole of their coal while others use their cleaning plants solely for contracts which carry premiums for cleanliness.

Table 6.—*Some Commercial Results*

Plant	Size, In.	Clean Coal Ash, Per Cent.	Fixed Ash, 1.5 Sp. Gr. Float.
F[a]	1½–5⁄16	5.00	3.90
G	1½–1	3.90	Average 2.55
	1–½	4.10	
	½–⅛	5.10	
	Size, mm.		
H[b]	10–3	6.15	3.65
	3–0.5	5.90	2.90
J[c]	35–20	3.15	2.50
	20–10	3.75	2.80
	10–2.5	2.85	2.80

[a] Plant F desires to make a clean gas nut coal and the results generally are practically on the ash point of the 1.6 sp. gr. float coal, loss of coal in shale being negligible.

[b] Plant H is treating a coal containing 5 per cent. middle product lying between 1.5 and 1.75 sp. gr. It is not possible to remove the whole of this but sufficient is taken out to give a desired product averaging 6 per cent. ash.

[c] Plant J represents results from a cleaning plant of much later construction than any of the others described in this paper, and is perhaps a reflection of the technical advances made in the art of pneumatic separation during the past few months.

Power costs vary considerably in addition, and, since power is an important item of cost, comparison becomes very difficult.

Thanks to the kindness of the colliery companies concerned, however, who have been at some trouble to prepare figures on a reasonably standard basis, it is possible to publish for the first time some very interesting figures as to operating costs and results.

In each case, the following points should be borne in mind.

1. No cost is included for loss of vend due to stone removal.
2. All costs are based on the raw coal treated.

3. Savings due to the operation of a plant have not been set off against the costs. Several cases are available where the savings (such as elimination of hand-pickers) are substantially greater than the operating cost.

4. No capital charges are included, since these vary very much from company to company, while the capital cost itself varies considerably in accordance with the amount of external handling plant installed for each individual installation.

5. No plant whose costs are included has operated for less than one year, while two have been in operation for nearly five years.

TABLE 7.—*Operating Data from Commercial Plants*

	Plant A[a]	Plant B	Plant C[c]	Plant D	Plant E[e]
1. Size of coal treated, in.	1½–0	1½–0	2–0	2–¼	2¼–0
2. Type of plant	Central. Four collieries	Colliery. One seam	Central. Two collieries	Colliery. Three seams	Colliery. Two seams
3. Class of coal treated	Coking	Gas and coking	Gas	Gas	Gas, coking and steam
4. Average ash in raw coal, per cent.	8.08	12.98	14.25	15–22	12–15
5. Average ash in clean coal, per cent.	6.54	7.04	7.75	5.00	6.20
6. Fixed ash (1.5 sp. gr. floatings), per cent.	4.00	4.70	4.6	3.80	4.60
Fixed ash (1.6 sp. gr. floatings), per cent.	4.50	5.45	5.15	4.10	5.20
7. Coal in stone (1.5 sp. gr. floatings), per cent.	3.91	2.5	2.1	0.8–1	2.00
8. Tons treated, per annum	515,824	141,465	286,926	337,500	453,735
9. Approximate tonnage treated to date	830,000	155,000	1,150,000	1,400,000	500,000
10. Electric current, pence per ton		1.14[b]	1.23[d]	1.07	0.84
11. Labor, pence per ton		1.31	1.19	0.78	1.38
12. Maintenance, pence per ton		0.34	0.67	0.11	0.49
13. Total operating cost, pence per ton	2.06	2.79	3.09	1.96	2.71
14. Number of separators	4	3	8	4	8
15. Sizes treated, in.	1½–⅝ ⅝–³⁄₁₆ (2) ³⁄₁₆–0	1½–½ (2) ½–0	2–1 1–½ ½–¼ ¼–⅛ –⅛	2–1½ 1½–1 1–½ ½–¼	2¼–¾ ¾–⅜ ⅜–⅛ –⅛

[a] This plant was designed to treat coking smalls with average ash content of 11.5 per cent. and on normal duty gave clean coal ashes which varied from 5 to 5.5 per cent. For some time, however, run of mine coal broken down to 1½ in. has been fed, giving a cleaner raw coal, while for special purposes the coal –⅛ in., amounting in quantity to 40 per cent. of the whole, has been by-passed uncleaned and remixed with the cleaned upper sizes, giving a clean coal ash of approximately 6.5 per cent., which is the desired figure at present. The high figure for free coal in stone is due to the inherent difficulty of treating an already clean coal without some such loss. The coal has a moisture content running as high as 6 per cent. Plant runs 22 hr. daily 7 days per week.

[b] 0.5 d. per kilowatt-hour.

[c] This plant has been in operation for several years and contains six small separators of an early pattern. The coals coming from each colliery differ substantially in characteristics and there is no regularity of mixture. Moisture content is high and varies considerably. 5 per cent. of dust containing 14 per cent. ash is remixed and included in figures of clean coal ash.

[d] 0.75 d. per kilowatt-hour.

[e] Dust removed from this plant is not remixed, but is burnt. Cleaned fines are taken for coking, other grades going for gas and steam-coal markets.

6. The expression "fixed ash" is taken here as the "float ash" of the coal in a 1.5 sp. gr. solution, since this is the commercial fixed ash commonly used in connection with this work. This fixed ash varies considerably in different sizes of coals, particularly in Durham, where certain seams have a float ash of 1.5 sp. gr. running up to 6.5 per cent. in the nut sizes and down to 3.5 per cent. in the fines.

7. "The free coal in stone" is the coal floating in a 1.5 sp. gr. solution. It is comparatively easy to produce a good figure for free coal in stone by using a 1.35 sp. gr. solution, but the author uses 1.5 for both coal and stone in order to have a standard basis for all work done.

8. The differences between the fixed ash and the clean coal ash vary from colliery to colliery. This is due to the fact that the plants from which detailed figures are given are treating coals containing a fair percentage of middlings but are removing only the pure shales heavier than, say, 1.7 sp. gr. There is thus left in the "clean coal" all middle products higher than 1.5 sp. gr., which accounts for apparent variations in efficiency of cleaning.

In order to indicate efficiencies when the above conditions do not apply, further figures are given, showing commercial results obtained from other plants where clean products are desired as near to the 1.6 sp. gr. figure as possible (Table 6). These are lettered F to J. It has not been possible to obtain the operating cost figures for these plants or any other information than that given.

Fairly complete details are available for five plants, which have been selected to illustrate varying types of installation and kinds of coal treated (Table 7). Unfortunately, it has not been possible to obtain complete details in every case but the figures published contain the exact information supplied by the colliery companies owning the plants. These plants are lettered A to E.

General Problems

A problem that urgently requires some general and economical solution is that of disposal of fine dust from the collecting system.

Figs. 13 and 14 illustrate a large plant capable of treating 250 tons of coal per hour. The dust collectors are at a little distance from the plant and actually adjoin a new boiler house containing three boilers with a capacity of 50,000-lb. per hour, fired by pulverized fuel.

This fine installation takes all the aspirator and filter dust and consumes it with a consequent considerable improvement in the quality of the gas and coking coals leaving the pit and also substantial economies due to the replacement of salable coal by dust for steam-raising purposes.

Briquetting and carbonization by low-temperature processes are alternative methods of using this difficult material, but some cheap and

easy use for it would take a load off the shoulders of many designers of modern preparation plants.

Quite outside the ordinary advantages to the metallurgical, gas and other industries which have been so often quoted—lower transport cost,

FIG. 13.—PLANT CAPABLE OF TREATING 250 TONS OF COAL PER HOUR.

greater throughput of coke ovens, easier working of retorts, etc., etc.—are a few which have shown up well in commercial practice in European plants as follows:

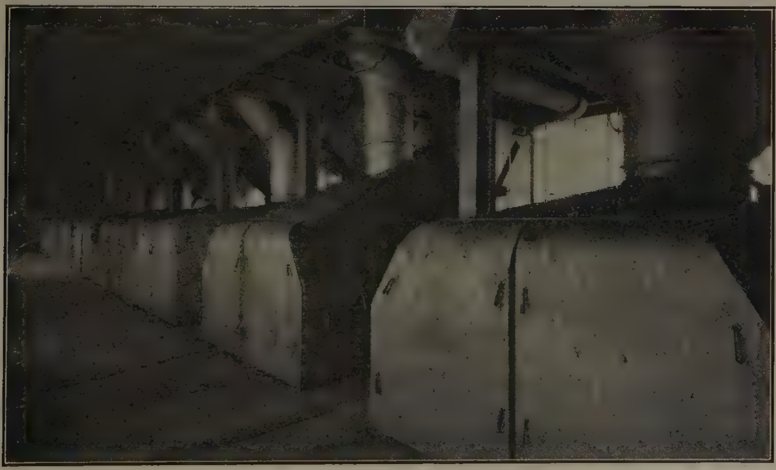


FIG. 14.—SEPARATOR FLOOR IN PLANT SHOWN IN FIG. 13.

1. During the bad times through which the coal industry has passed in the past few years, collieries operating dry-cleaning plants have not lost any time at all, and this capacity to work full time has shown the greatest financial return for the expenditure entailed.

2. Without in any way reflecting on wet processes, it can be truly said that a dry-cleaned coal, well prepared, almost always looks better than a washed one, and it has been found on many occasions that the good appearance of the coal has had as much effect on the buyer as the low ash content.

3. Coal does not freeze in railroad cars if cleaned dry. Frost does not offer a great deal of trouble in England as a rule, and most wet washeries keep going. The severe winter of 1928–1929, however, caused much dislocation of traffic on both rail and sea, owing to freezing of consignments in cars. This is a normal Continental trouble which dry cleaning eliminates, as was proved during the severe winter referred to above.

4. Very considerable reductions in costs generally have been made by some companies working seams containing soft bands. These soft bands disintegrate in a wet washer and cover the coal with a gray film, thus spoiling its market appearance. Normally, therefore, they are cut out under ground where possible, but in the instances referred to this practice has been changed. The band is now loaded out with the coal and dry-cleaned out at the surface, with a largely increased output of good coal per man as a result, due to the easier extraction possible.

5. Finally, experience has conclusively proved that dry-cleaned coal, particularly in gas and coking markets, commands better prices; so much better, in fact, that instances are within the author's knowledge where the increase of prices and clean coal bonuses obtained, together with the general economies due solely to the plant, have actually been sufficient to pay for the installation completely in approximately three years.

Coal-cleaning plants in Europe are expensive. As a rule, their capital cost is in the region of £220 per ton per hour, but if carefully designed and properly operated by selected men who are keen on their work and have plants to be proud of, they do well for the industry and are a source of profit to the people who invest their money in them.

DISCUSSION

(*Chester M. Lingle presiding*)

H. N. Eavenson, Pittsburgh Pa.—I do not want to go into the technical part of the paper. When in England it was my good fortune to spend some time with Major Appleyard, and we visited a couple of the plants described. I have no hesitation in saying that as far as the arrangements for taking care of the dust and for taking care of the machinery generally were concerned, they were far in advance of anything that I have seen in this country.

T. Fraser, Pittsburgh, Pa.—One of the noteworthy points of Major Appleyard's paper is that he dealt so much with the auxiliary things rather than with the development of the cleaning machines themselves.

The problems met with and apparently solved effectively are the same as those met with in the practice of dry cleaning in America. We ought to be interested in the data he gives on aspiration. That has been tried to some extent in America,

but it has not found much favor. After reading this paper, probably I would say that the reason is that it was not followed as persistently as in England, and we have not tried as well designed an aspirator as they are now using. I wonder if that might not also be due to the difference in the water content of the coal?

What are the limiting water contents that can be handled with these aspirators? As a rule, do the coals that are treated in these plants vary as greatly in water content as ours do, and do they occasionally have to handle coal that is dripping with water? In such cases, has heated air been tried in the aspirators?

K. C. Appleyard.—Possibly one of the reasons why aspiration has not been pursued in America is the fact that it is not necessary. We do not always aspirate in England; it is not necessary where the ash content of the dust is not greater, or is very little greater, than the ash content required in the clean coal. When dust runs up to 30 per cent. of ash, we find it necessary to take it out.

In regard to the point at which moisture content begins to defeat an aspirator, it is rather like asking where moisture content begins to defeat a pneumatic separator. It is very difficult to generalize. We are aspirating coals where the moisture content in the fines below ⅛ in., or 3 mm., runs as high as 12 to 14 per cent., but one cannot pretend that the efficiency of aspiration is as great as with a dry coal.

So far as treating dripping wet coal is concerned, we do not tackle those propositions dry. We have not tried the use of heated air, although it has been considered from time to time. So far we have operated without it, but I am of the opinion that on Continental work we shall shortly come to consider some method of passing heated air through either the aspirators or the machines, or of driving off moisture in some such way before it goes to the plant, but we have not reached that stage yet.

F. A. Jordan, Youngstown, Ohio.—I think everyone will agree that the handling of the dust, whether it be by a wet or dry process, is a difficult part of the problem of cleaning coal. However the necessity or advisability of aspirators in connection with a dry-cleaning plant is not entirely clear to me. The air from under the table lifts the same dust, I would say, that the aspirator would have removed.

It seems to me that the table aspirates the coal and takes out the greater part of the dust, at least it can be made to do so. We enclose the tables and provide for taking care of the dusty air, and I would say that, when the tables are so equipped, we could lift the dust in the same way as the aspirator, and also any additional dust that comes from degradation of the coal as it is moved along in its travel on and ahead of the table. Surely we can lift dust up to the saturation point of the tables' quantity of air. Why not take all the dust out with the table rather than add to the equipment in the plant?

K. C. Appleyard.—The reason one aspirates, as a rule, the reason we aspirate in Europe before cleaning, is that it is impossible to remove the dust from a coal while it is on the cleaning table. The pneumatic separator is designed to take the stone out of the coal and it is not designed to take out the dust. The bed of coal, without its dust, is more fluid and more susceptible to treatment and less liable to give fluctuations in the clean coal ash when the dust is out of it than when the dust is in it.

I admit that it sounds like doing the work twice. First to aspirate it out and then to have a dust-cleaning system, but, as is rightly pointed out, there is a certain amount of degradation as the coal passes through the flow sheet, in bunkers and as the coal moves over the pneumatic separator. Unless there is some method of cleaning that or of taking it away, it is bound to be a considerable nuisance.

Certain plants in England and in Europe are working without aspirators. It is not, by any means, uncommon, and, as I pointed out, where the dust ash content is no greater than that desired in the clean coal, we never do aspirate, but we also begin to

suffer then from dust troubles. With the rather greater volume of air coming through the machine at the feed end, much of the dust is blown just off the table and drops round underneath it, presently building up until the connecting rods of the machine are running in dust; and although one is prepared to put up with that to some extent, as a rule it is not desirable. The additional cost of providing an aspirator complete is not much more than $1000 or $1500, and we think it worth while, as a rule.

R. W. Arms, Chicago, Ill.—I think Major Appleyard and his company, the Birtley Iron Co., of which (I happen to know personally) he is the inspiring and energetic leader, are to be congratulated on the accomplishments made in England in air cleaning.

The saying is that a problem well stated is half solved, and in Major Appleyard's paper he has stated his problem, in the beginning, as it existed in 1925, and, to a certain extent, as it exists today not only in England but in every other country.

One important advance that is being made is the reduction of air consumption. In the beginning when the individual sizes were treated, especially the larger sizes, on separate tables, there was a tremendous amount of air used per ton of coal cleaned. It would seem hardly possible to make an 80 per cent. reduction, but we are glad to see that this reduction has been accomplished.

Similar attempts are being made in this country, and perhaps, although we look at certain of these problems as laid down by Major Appleyard as being more important than others, nevertheless some effort is being made on each one of them.

In connection with the discussion which has just taken place between Mr. Jordan and Major Appleyard, I might add that perhaps one reason why we have not gone to aspiration in the United States is because we have looked at the problem from a slightly different angle and have been attempting to clean fine coal without aspiration, by the use of the so-called high-pressure decks.

We have thought that a material could be supplied as a deck surface which would back up a tremendously high pressure compared with any pressures known today. This material may be punched metal, or screen cloth, or some kind of a fabric. If a material could be found which would back up this pressure and supply, to a relatively thin bed, a definite quantity of air regardless of the amount of moisture present, regardless of the amount of fines present and regardless of the screen analysis of the coal that might be on the table, as perfect a separation as could possibly be made would be effected.

We have not found that ideal substance as yet, although we improve as we go along. However, this search is perhaps the reason why we have not selected aspiration as the proper solution.

Along the lines of dust collection we have done similar things. We have used cloth in the form of bags or sheets, preceded by the substitute for a cyclone, which is an expansion chamber or some place where the expanded air can drop its burden of larger dust.

I am glad to hear Major Appleyard speak about presizing and, of course, I am highly pleased to hear him make such reference as he has to the Arms screen.

J. Griffen, Pittsburgh, Pa.—I also had the great pleasure of a visit with Major Appleyard about a year and a half ago. I spent several days looking over the various plants, and I was impressed by the thought that has been given to the design of plants, not only so far as cleaning is concerned, but in the handling of material, design of structure and elimination of dust nuisance.

This paper shows that the problems involved in the application of dry cleaning to the coals of England—they were the only plants I saw—have been thoroughly studied, and the Birtley Iron Co. organization has done an enormous amount of effective work. I am glad to note that there have been definite improvements on some steps which,

at the time I was there, were considered almost perfect—that organization is not satisfied with a thing which apparently is a solved problem.

I also spent some time in the manufacturing shops. I was much impressed with the great care in the shops in the construction of the actual air tables; the care with which all parts were assembled. The resulting machine is a beautiful one, infinitely better looking than one would expect to find around a coal-washing plant.

I would be interested to know the amount of steel per cubic foot of building structure required to produce a rigid structure. That might be of some interest to us in connection with our steel construction in this country.

Another point I noticed is that in England they are not particularly interested in mechanically cleaning, either wet or dry, coal much larger than 2½-in. dia., whereas in this country we seem to be interested in coals up to 4-in., or perhaps 6-in. dia. Perhaps for that reason the Birtley Iron Co. has not been interested in developing machines of supercapacity, you might say, for handling mixed sizes. In other words, pneumatic machines have been developed in this country which, it is claimed, will handle upwards of 250 or 300 tons an hour, of 5 or 6 in. to 0.

Generally speaking, with English bituminous coal the amount of refuse is considerably higher than in our bituminous coals. I believe it is true that County Durham coals run 10, 12 and 14 per cent. refuse on an average, perhaps higher than that at times. That probably influenced the design of the V-table as against the Y.

County Durham coals, particularly, are rather dry. I wonder what variation, or what effect on the uniformity of the product, variation in the moisture content of the raw material may have.

K. C. Appleyard.—It is a little difficult, offhand, to say how much steel there is in a building, but I have in mind now a particular plant where the weight of the structure was approximately 1 ton of structure per ton of coal per hour. It was intended to treat 250 to 300 tons of coal per hour, and I believe that actually it had 257 tons of steelwork in it.

We have not, in Europe, been faced with the problem of large capacities. From the study that I have made of problems in America, and my American friends whom I see in Europe, there appears to be a pressure here for a large unit that will treat unsized coal, and how it treats it I do not quite know. What the result is, I do not know, but the general idea is to have one large unit.

There is, I think, no difficulty whatever about providing these large units. Colliery owners and coal operators will have to make up their own minds as to whether they are worth while. I feel satisfied that if you start feeding −6-in. coal into any kind of a system, and particularly to feed it dry into any kind of a system such as we are considering, the amount of degradation will be large with any friable coal.

I did, however, have an argument the other day with an American friend who quoted a large machine to me, and I asked him to work out its capacity per square foot of deck area. That is the point that matters. What is the capacity per square foot of deck area? The large machine he had in mind had a capacity of 1 ton for every 2.4 sq. ft., as against the machine that I have been trying to describe to you of practically 2 tons per square foot, the difference in capacity being about six times.

There is no real reason why capacity should not have been multiplied considerably further than we have yet gone, but so far, in Europe, there has been no demand for it.

In regard to the question of moisture, I think we showed Mr. Griffen a plant in which the average moisture content ran about 6 per cent. It varies from 2.5 to 6 per cent., and up to that point the differences in the result are not noticeable. However, I must point out that this moisture content is the total moisture content. The

free moisture content in this particular plant is about 3.5 to 4 per cent., and when it gets, with this particular coal, above 5.5 per cent., we begin to be affected. Below 5 per cent. moisture it has very little effect.

It is impossible to generalize about moisture. I have known cases where 2 per cent. of moisture has completely defeated a pneumatic separator. I have known other cases where a pneumatic separator has taken up to 15 and 16 per cent. of moisture and dealt with it easily.

C. M. Lingle, Nemacolin, Pa.—Mr. Jordan, in reference to Major Appleyard's remark on the capacity of the tables, I think you know of the table that was reduced without impairing either its capacity or efficiency. As I remember, it was reduced about 40 per cent. in its length?

F. A. Jordan.—The chairman has called on me rather suddenly for some figures that have almost slipped my memory, but the thing that he has in mind, I think, is a dry table of rather large capacity. The table was built rather long, so long and so heavy that it gave mechanical trouble. To overcome the mechanical trouble, the table was shortened about 30 per cent. of its entire length; that is, when it was completed, it was 70 per cent. of its original area.

Of course, the weight of coal that reduction represents would not help very much to overcome the mechanical difficulties because that was a small part of the total weight. But stripping the deck and relieving the structure of all the superfluous steel had its effect upon reducing the weight.

I think Mr. Lingle has in mind that when we reduced the number of square feet of table top, much to our surprise, we did a little better job of cleaning. We overcame, very largely, the mechanical trouble, and we increased the tonnage from about 190 tons to about 210 or 215 tons per hour. So I think probably there is much to be learned about the number of tons per square foot of table that can be handled.

K. C. Appleyard.—I was not trying to start an argument. I was just trying to make the point that the standard of comparison, the standard of judgment of these matters, should really be the effective capacity per square foot of deck area of a machine. It is the only basic comparison, if you keep the same standard of efficiency so far as cleaning is concerned. That is the whole point that I desired to make.

C. M. Lingle.—Well, Major, I thought it would be interesting to you to know that we buy things here very much too large and cut them down.

E. O'Toole, Jr., Welch, West Va.—We have done considerable experimenting in cleaning coal dry on our separators. We have progressed from 10 tons per hour, when we first started, to almost any quantity up to 125 to 150 tons per hour, depending upon the size and kind of coal treated. We have changed our practice from prescreening to mass treatment and we now have three general kinds of cleaning which we call Simple Cleaning, Difficult Cleaning and Extraordinarily Difficult Cleaning.

Simple cleaning is used where a resultant ash in clean coal of only 7.5 to 8 per cent. is desired, or when the extraneous dirt that is to be removed is not included in the coal below $-\frac{1}{4}$ inch.

Difficult cleaning is when the coal treated contains a large percentage of extraneous material between $\frac{1}{4}$ and $\frac{1}{16}$ in., which must be removed to meet the demands of those using the equipment.

Extraordinarily difficult cleaning is when the large percentage of the coal contains an unusual amount of high moisture and extraneous material to be removed below $-\frac{1}{16}$ inch.

To meet these difficult cleaning problems, we have designed and applied for patents on various modifications of our separator, for the purpose of giving the desired stratification of the material treated, and continuous feed that can be adjusted when the separator is in operation; also chutes that return the middlings to the separator without degradation.

We have found that engineers on wet coal preparation have given little attention to the nondegradation of the fines. They have called $-\frac{3}{4}$-in. coal slack and have not considered degradation of coal of this size a serious consequence, but we find it very necessary to prevent degradation of this size of coal on account of the large amount of dust that is created, which must be separated from the air, but more particularly on account of the degradation of the refuse that is in this fine coal and must be separated from the coal.

We have adopted as a cardinal principle the nondegradation of the refuse mixed with the coal, as it is the fine refuse that is difficult to separate from the coal.

To illustrate, we will take a hypothetical plant as usually designed for wet coal preparation when it is not desired to wash the $-\frac{3}{4}$-in. coal on account of the water it would absorb, or when possibly the $-\frac{3}{4}$-in. coal is to be treated by the dry method.

All of the coal is placed on a shaker screen, the $+\frac{3}{4}$-in. is taken out and the $-\frac{3}{4}$-in. is collected on a back-action pan or chute which delivers it to a dragline conveyor, which delivers it to the boot of an elevator, which elevates and discharges it into another dragline conveyor, which delivers this coal to the hoppers from which it is loaded into railroad cars or to dry-cleaning separators. The back-action chute has caused degradation, the dragline conveyor that conveys the coal from the back-action chute to the elevator boot has caused degradation, the buckets of the elevator when picking this coal up out of the boot have caused degradation, the dumping of the buckets at the top of the elevator has also caused degradation, the dragline distributing conveyor that delivers the coal to the loading bin or surge hoppers has caused degradation, while the fall from the dragline conveyor to the surge hoppers or loading bins has also caused degradation, with the result that the coal in the surge hoppers or loading bins bears very little relation, regarding size of coal and the size of the refuse mixed with it, to that first received on the back-action chute.

Now we will suppose that the coal and refuse that is in the surge hoppers is fed to and passes over dry-cleaning separators. This is the first place that the coal in its travels will receive very little degradation, the coal is carried on the air, the refuse is partly carried on the air, when discharged from the separator the refuse passes into the refuse chute and is no further trouble to anyone, the middlings which is part coal and concentrated refuse drops into another dragline conveyor which discharges into an elevating conveyor, which drags it up a 30 per cent. slope and discharges it into a screw conveyor, which discharges it into the boot of the crude-coal elevator with the crude coal and from there it is recirculated with the crude coal as described above. This description shows the necessity for astute coal-preparation engineering.

With our separators we find no difficulty in making a highly efficient separation of the coal and refuse contained in it on sizes between 3 and $\frac{1}{8}$ in., hence we call this simple cleaning. It is the $-\frac{1}{4}$-in. mixtures that are difficult and extraordinarily difficult, as stated above. Highly efficient separation of the larger sizes can be made by the wet method, but the wet method finds separation very difficult on the $-\frac{1}{8}$-in., and most wet washers make no attempt to clean this size of coal, on account of the large percentage of water this coal will retain over a period of time.

We are making the claim that there is no longer a place in coal preparation for water. Although the practice of wet-washing coal has been in vogue for about 100 years, it has been wrong for about 100 years.

J. B. Morrow, Pittsburgh, Pa.—There are just one or two points I would like to ask about. What size of coal was used in plant J?

K. C. Appleyard.—The smallest size is 2½ millimeters.

J. B. Morrow.—Was that coal all cleaned at one size?

K. C. Appleyard.—No, the coal was cleaned from 35 to 20 mm. in one size, 20 to 10, and 10 to 2½. They were cleaned in those sizes because those are the market sizes.

J. B. Morrow.—From our standpoint the most important comparison between wet and dry cleaning methods is in the cleaning of the fine sizes. For instance, in one of our wet plants a combination screening and cleaning process will give us a coal from 4 in. to 100 mesh, which is within 0.1 to 0.2 per cent. of the inherent ash in the raw coal at 1.40 sp. gr. Under this same condition there will be approximately 0.5 sink at 1.60 sp. gr. in the 4 in. by 100-mesh coal. Our experience on dry cleaning has been that the efficiency falls off rapidly below 14 mesh and for high-grade metallurgical coals this degree of cleaning is not sufficient for the present exacting markets. Has any work ever been done on cleaning coal from 1½ mm. down to ⅕ mm.? When cleaning −5⁄16-in. coal on one table, about how far down would you expect to actually get efficient cleaning? Would it be at ⅛ or 1⁄32 in.?

K. C. Appleyard.—Generally speaking, we do not size under any circumstances in England below ⅛ in. We treat −⅛ in. in one size. Down to about 1⁄32 in., the cleaning is practically perfect with dry coal. It begins to fall off below 1⁄32 inch.

I am interested to hear, for the first time, that an ordinary wet-washing process is able to wash coal down to this very fine size without any loss of efficiency. Up to date I have never seen it done. The Germans tried to do it for a considerable time, and the French and the Belgians, but the only process that I know of that has done it satisfactorily below 1⁄32 in. is the flotation process, and if it is really being done effectively below 1⁄32 here, I should like to see it.

J. B. Morrow.—I shall be pleased to show Major Appleyard where coal is being cleaned at high efficiency down to ⅕ mm. and we might carry it a little further and show him some interesting figures down to 100 mesh. On oil flotation we have done good work down to 200 mesh, but not as well in the −200 mesh.

However, this does not mean that it is impossible to clean coal from 4 in. to 100 mesh in one unit. At least a part of the fines will have to be taken out and recleaned. I do not believe it is possible to obtain efficient cleaning in any unit with from 4 in. to 100 mesh without any re-treatment.

As I see it, a large part of the dry cleaning problem is to get some method that will efficiently clean coal down to ⅕ mm., giving a clean product −4-in., or 2½ in. by ⅕ mm. that will be within less than 0.5 per cent. of the inherent ash in the float coal. If we ever reach that stage, I have no doubt that dry cleaning will supplant the wet methods, but so far, to the best of my knowledge, this has not been done.

K. C. Appleyard.—We have never been asked to undertake this fine cleaning problem right down to the last degree, There is pressure being put upon us now to do so. We are at present designing a plant to treat 200 tons of coal per hour below 3 mm. That is a bit of a problem, especially with a slightly sticky coal, but I am satisfied that by re-treating that coal on the present type of machines we can get a very high efficiency.

Combination Wet and Dry Coal-cleaning Process

By Ray W. Arms,* Chicago, Ill.

(Pittsburgh Meeting, September, 1930)

The combination wet and dry coal-cleaning process is not an attempt at a compromise between the wets and the drys nor is its merit confined entirely to the limitation of moisture in the smaller sizes. It is more correctly, and more broadly, an application of the sized-coal cleaning principle adapting itself in an ideal fashion to present day multiple-size preparation practice.

The purpose of this paper, therefore, is to call attention to the broader significance of the term "combination wet and dry process" and to mention some of the factors that govern its application.

In the narrow sense, when considered apart from the sized-coal cleaning principle, the combination process finds its justification chiefly in the ability to limit the moisture in cleaned coal. Larger sizes of coal offer less superficial area compared with their bulk for the attraction of moisture films. They drain more readily in bins or railroad cars, having more interstitial space for the free flow of water. Therefore, they neither attract nor hold objectionable moisture percentages. In fact, there is so keen a demand for certain washed sizes containing some moisture that hygroscopic salts such as calcium chloride are applied to attract and retain it.

When coal deliveries are made on a guaranteed moisture percentage the coarser sizes may be washed and mixed back with the fines without adding greatly to the original moisture but the fines must be dry cleaned or dried to control moisture. Thus in all respects moisture in the coarser sizes is less objectionable than in the fines. The implication of this statement, that moisture is always more or less objectionable, is true. It is safe to say that if dry-cleaning apparatus could be perfected to a point at which its technical results, cost and other aspects would be equal to wet washing, water would no longer be used. As it is, we find that good technical results are more difficult to obtain by dry cleaning than by water. Obviously the use of air tables for coarse coal is not ideal, for it involves the attempt to stratify material in a thin bed on a table surface when the load is only one particle deep. Furthermore, the various air cleaning tables on large sizes require a large volume of air for separation, more power to operate, more dust-collecting apparatus and involve more coal breakage.

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Moisture in fine coal (slack), on the other hand, is universally objectionable. Being as inert as slate, it is nearly as important as a fuel dilutant, especially as fine coal retains high percentages of it, even after long draining. In fine coal it freezes in railroad cars, produces various undesirable effects in coke ovens, adds greatly to freight and is generally undesirable. The fact that under certain conditions a dry and dusty coal is undesirable and water is added in the form of sprays does not necessarily condemn the dry process, as obviously it is simpler to add water than to remove it.

For many purposes moisture up to certain limits is desirable. With dry-cleaned coal exactly the desired amount of water can be added at the proper time, avoiding payment for freight and the difficulties of handling wet coal.

Sludge handling, water clarification, filtering, centrifugal and heat drying and other disagreeable operations are avoided by dry cleaning.

The technical results obtainable by dry cleaning are fairly well known by this time. When all conditions are propitious, that is, when the coal is dry, refuse not too flaky, bony and middle gravity material at a minimum, etc., the results of dry cleaning are as good as the most complicated washing process. Under these conditions the advantages of dry cleaning outweigh those of wet cleaning. However, moisture in the original feed from the mine affects dry cleaning more or less, and so also do other variables. When choosing the dry process these difficulties and the results possible with the coal under consideration must be weighed against the possibly greater ash reduction of a wet washer with its attendant difficulties of sludge recovery, drying, and so forth.

Division between Wet and Dry Cleaning

The reference to "coarser" and "finer" sizes demands a definition. The ideal dividing line between wet and dry cleaning is approximately ½ in. It is below this size that moisture attaches itself to coal particles in rapidly mounting percentages. It is on coal above this size that air tables require increasing quantities of air. This dividing line should be studied in every case, however, for it will be affected by lack of water in the vicinity of the cleaning plant, by the selected compromise between ash reduction and permissible moisture, and by many other influences.

Evaluation of Moisture Percentages

Attempts have been made to put moisture on some definite basis and evaluate percentages that "cause freezing in railroad cars," or "cause screening difficulties" or "are ideal for coking," or produce numerous other effects. To this end research has been energetic but ineffective. Moisture percentages that give rise to the various effects, injurious and otherwise, depend on so many variables, such as physical nature of the

coal, inherent moisture, methods of moisture analysis, and so forth, that no definite moisture percentages can be given for any of the well-known phenomena to which moisture gives rise.

Some attempt was made recently to plot the moisture added by washing coal against the size of the coal. Data were obtained from about 150 washing tests from various parts of the country. As each test was made under exactly the same conditions and samples were taken and analysed under identical circumstances, it was hoped that the shape of the curve would be illuminating. In spite of the fixing of some of the variable elements, the results were not conclusive because of the variety of coals treated, but sufficient evidence was produced to indicate the following approximate figures, which will give some idea of the moisture retained by a nonporous coal, such as the Pittsburgh seam, after washing and draining on a draining screen:

Size, In.	Excess Water Retained, Per Cent.
4×2	0.7
2×1	1.5
$1 \times \frac{1}{2}$	2.2
$\frac{1}{2} \times \frac{1}{4}$	4.0

Below $\frac{1}{4}$ in. the moisture percentages were too variable to report.

Cleaning Sized and Unsized Coal

To combine wet and dry methods in one cleaning plant necessarily involves presizing. The cleaning of sized coal differs fundamentally from unsized coal cleaning. In the latter category may be included all processes and self-contained machines, wet or dry, which operate on a feed characterized by a large range of sizes such as 4 to 0 in., 2 to 0 in., etc. Small slack itself is an example of unsized feed. Even though dedusting by air or screens is attempted, the feed is essentially unsized, as the distinguishing characteristic of the unsized feed is that many sizes are cleaned simultaneously in a common apparatus. Whatever subsequent re-treatment any part of the original feed receives does not remove it from this classification. Middlings or refuse product may be re-treated or the fine coal may have a separate rewash but the primary process is designed for unsized feed. The best known American machines which occasionally or consistently operate under this principle are Rheolaveur, Peale Davis table, Chance sand flotation, various types of jigs and wet cleaning tables.

Sized-coal cleaning, on the other hand, is the principle on which some equally important machines and processes operate. These receive the coal in an essentially presized condition, the nature of this presizing depending either upon the market grades of coal desired or upon the cleaning possibilities of the process. Sizing may go far beyond the demands of the market to accommodate the process, and certain sizes

screened for cleaning purposes may be reassembled for shipment, or the requirements of the process may fall short of the market demands and some screening may be required after cleaning. In sized-coal cleaning the significant fact is that coal is screened within some definite and restricted ratio of sizes before being subjected to the cleaning apparatus.

Various types of jigs, pneumatic tables and hydroseparators may be used for cleaning either sized or unsized coal, but certain makes are more or less definitely committed to the cleaning of sized coal.

Ratio of Sizes

By "ratio of sizes" is meant the relation between the diameter of the mesh through which the coal has passed to that on which it has been retained. Thus, 4 by 2-in. egg coal is screened on a ratio of 2 to 1 and a mixture of egg, nut and pea through a 4-in. screen and retained on a ½-in. screen has a ratio of 8 to 1. This nomenclature applies to either round or square holes but some adjustment of equivalents should be made if the larger holes are of different shape from the smaller holes.

"Ratio of sizes" is a term used either to designate the proper prescreening as preparation for the sized-coal cleaners or to indicate the effective range of an unsized cleaning device. In either usage there are several contributing factors, which will be discussed briefly.

Many writers have stressed the influence of free settling and hindered settling on the ratio of sizes. The two principles have been recognized for many decades and it is not the purpose of this discussion to amplify the many theoretical discussions and formulas pertaining thereto. However, the definite mathematical nature of the expressions of these principles leads one to believe that a cleaning device must operate definitely and exclusively according to one or the other. This is distinctly not true, as there are many phases of each and a process can operate according to both at the same time, or, more properly, according to a combination of the two principles. The operation of one machine may tend strongly towards hindered settling, whereas another may depend more definitely on free-settling activity for its success but still derive some benefit from the greater selective powers of the hindered-settling principle.

"Free settling" and "hindered settling" usually involve the idea of uprising currents of water or air in their manifestation, but coal cleaning makes use of other equally important accessory principles. Agitation in reduced upward currents accomplishes a stratification according to gravity and size entirely different from the stratification in a so-called hindered-settling classifier depending on upward current alone. The wet cleaning table gives excellent evidence of this effect, although its operation is not confined to this principle alone. Agitation at high frequency, or "vibration" in reduced air currents, has not as yet been

extensively applied, but there are some interesting experiments under way, which may lead to developments in dry cleaning.

The principle involved in the flowing stream, which causes material to stratify without the use of rising currents, is best exemplified in the launders of the Rheolaveur. However, wet tables combine this effect with agitation to produce results and dry tables use stream flow together with rising air currents and agitation.

It is evident from the preceding discussion that theoretical formulas cannot be depended upon to establish the proper ratio of sizes for any cleaning process. The intelligent balance between high-grade technical results and some degrees of improvement at a low cost has always governed the ratio of sizes used in sized-coal cleaning processes. Technically the nature of the coal and its impurities affects this ratio, as the more difficult the coal is to clean, the smaller the ratio should be.

By far the most important influence on the ratio of sizes in a sized-coal cleaner is the demand of the market. If the output is to be 4 by 2-in. egg, 2 by 1-in. store and 1 by ½-in. pea, those are the sizes used in prescreening regardless of the theoretical possibilities of the machine. However, as capacities of the units and tonnage to be cleaned must be considered together with the ultimate cost per ton in the design of a cleaning plant, it is well to mention certain ratios, that govern the feed of certain kinds of equipment.

Various types of jigs have used all conceivable ratios, but with hindered-settling principles a 5 to 1 ratio is practicable for good cleaning. Pneumatic tables have operated largely on a 2 to 1 ratio but the present tendency is to use wider ratios, especially with fine coal, which is difficult to screen. The Hydroseparator is the best example of the sized-coal cleaner. It has been devoted almost exclusively to sized-coal cleaning. When carelessly operated it acts under free-settling conditions but niceties of adjustment and skill in operation will cause it to retain a bed in a thick fluid condition just short of "freezing" and thus introduce hindered settling to a greater and greater degree into the cleaning principle. The rule of thumb ratio for the Hydro is 4 to 1, although this rule is frequently violated to suit the specific needs of each case.

Cleaning and Recleaning

Cleaning plants are rapidly becoming more complicated and will continue to develop along lines of more complex flow sheets as ways and means are discovered to supplement the process without increased cost per ton. In the effort to approach the perfect cleaning of the washability curve, various portions of the cleaning-plant burden are washed and rewashed, crushed and circulated and washed again. This development in modern improved practice does not in any sense militate against the sized-coal cleaner. On the contrary, it offers an added incentive to

use this principle. In most instances the preponderant bulk of the feed is light, low-ash coal mixed with a small percentage of slate, bone, laminations and other impurities.

With the sized-coal cleaner it is possible to deliver most of the clean coal at once as finished product, reserving for a much smaller unit the task of rewashing and re-sorting the remaining, questionable ingredients.

Reasons for Sized-coal Cleaning

The underlying reason for cleaning coal by any process is to remove its impurities as far as possible, so as to render it a better fuel, and many phases and aspects of this general reason have influenced buyers of coal-cleaning equipment in the past.

The reason for cleaning coal for metallurgical purposes is well known, and need not be discussed here. Many cleaning plants are producing coal, for the owners or for general sale, that is to be used in the manufacture of low-ash and low-sulfur coke, and for this work the selection of sized or unsized cleaning may be made freely.

One operator built a very well equipped cleaning plant merely to sustain the reputation of the company for producing clean coal. The cleaning plant produces a relatively small ash reduction, but allows more and more coal to be shipped on large and exacting contracts. The combination wet and dry process was used to limit the moisture in the resulting product.

Sometimes run of mine coal is greatly improved by removing a certain small size range containing more than its share of impurities, cleaning it and returning it to the run of mine coal. This is especially true in the Miller seams of central Pennsylvania, where impurities occur from 4 to ½ in. This is an excellent example of sized-coal cleaning.

In the Pocahontas field, among others, small slack frequently is low in ash, but nut and pea coal are very dirty. Without a cleaning plant nut and pea coal must be shipped with slack to a slack market. By cleaning, not only are these sizes made saleable, but the smaller slack is automatically improved.

Hand-picking of egg coal is highly selective, but often expensive. Many egg-coal cleaners are now installed of which the sole justification is the saving in the cost of hand-pickers.

In some fields where domestic coal is washed, the buyer demands washed coal regardless of ash because of its good appearance. Some installations have been made largely because of this fact.

Certain small sizes of coal can be freed from fine slack only by washing, because of the difficulty of fine-screening wet coal. This has been an important reason for many installations of sized-coal cleaning equipment.

Actually, at a great many mines presizing is already accomplished and nothing but a sized-coal cleaner is used. Many mines now make lump, egg, stove, nut, small nut, pea, slack and other sizes. To apply an unsized cleaning process would mean that much of the present screening equipment would have to be discarded. Furthermore, often it is either not necessary or desirable to clean all of the sizes that are being made. Here the sized-coal cleaner finds an ideal application.

Undoubtedly scores of similar reasons influence the individual buyer, but those listed will serve as illustrations.

Conclusions

The combination wet and dry process and sized-coal cleaning is not offered as a solution of all coal-cleaning problems. On the contrary, it is recognized that the processes of cleaning both sized and unsized coal have their proper applications. However, it is hoped that some problems of coal cleaning may be clarified so that fewer mistakes will be made in the selection of equipment. By first determining whether a sized or an unsized cleaner is required, much of the uncertainty in the selection of equipment will be removed. The large variety of cleaning systems and machines on the market and the apparent transition from one accepted process to another so bewilders the prospective purchaser that he postpones the issue in the hope that one of the competing processes will so outdistance the others that no doubt will be left as to its superiority. Because of the varying conditions of the coal as mined it is extremely unlikely that any one machine now generally known will ever be able to meet them all. Perhaps someone will introduce a process so flexible in its application that it will become universal, but until then the machine and process best adapted to the specific needs of each plant must be selected.

DISCUSSION

(*Chester M. Lingle presiding*)

R. W. Arms.—In my paper I said that certain small sizes of coal can be freed from fine slack only by washing, owing to the difficulty of fine-screening wet coal. This may seem to be in contradiction to the statement by Major Appleyard that many improvements have been made in the dry screening of small sizes of coal. Under the conditions referred to, this is not a contradiction to his statement, as it is recognized that these improvements have been made. However, it is difficult completely to remove the fine coal which adheres to the larger pieces in a wet slack, and this condition, especially in the Pocahontas field, is responsible for a marked preference for washed coal, and this washed coal often demands a premium in the market over the equivalent dry-cleaned sizes.

J. R. Campbell, Scottdale, Pa.—On page 269 is a tabulation showing the added moisture on the Pittsburgh seam of coal or similar coals. I can say, from my experi-

ence, that Mr. Arms' figures are as nearly correct as it is possible to get them. We usually figure that +3/8 up to 4 in. will average about 3.5 to 4 per cent. total moisture.

Mr. Arms has not mentioned that it is possible to dry reasonably well the wet-washed coal, 1/4 in. by 0, by means of mechanical dryers. Mechanical dryers have been used in this country for many years, and the latest development of dryers shows that it is possible to dry wet-washed coal, 1/4 in. down, or 3/8 in. down to about 5.5 to 6 per cent. moisture if the fines are not excessive. By fines I mean 48-mesh by zero material. So that you could figure, by adding the usual percentage of 3/16, or 5/16, or 3/8 material, you should be able to allow 4.5 or 5.0 per cent. total moisture in the wet-washed coal, 4 in. to 0.

I can also confirm Mr. Arms' remarks that in the Pittsburgh seam there is a large percentage of low-gravity coal. Ordinarily 75 per cent. of the coal will float at a low gravity, which is usually low ash and low sulfur. The same holds true for the lower productive measures: A surprising amount of good coal at low gravity, the lower productive measures as represented by the Kittanning group of coals in Central Pennsylvania. The bone fraction in these coals is 15 or 20 per cent. In the Pittsburgh seam of coal the bone fraction varies from a small amount up to, in one case I know of, as high as 40 per cent., which would mean that the metallurgical coal or ow-gravity coal would be only 60 per cent. of a shipped product.

Mechanical Preparation of Pocahontas Coals—Some Factors in the Problem

By J. R. Campbell,* Scottdale, Pa.

(Pittsburgh Meeting, September, 1930)

During the past few years, the writer has had occasion to take several excursions into the realms of the washability of beds 3 and 4 of the Pocahontas coal and the proper handling of these coals in prepared sizes

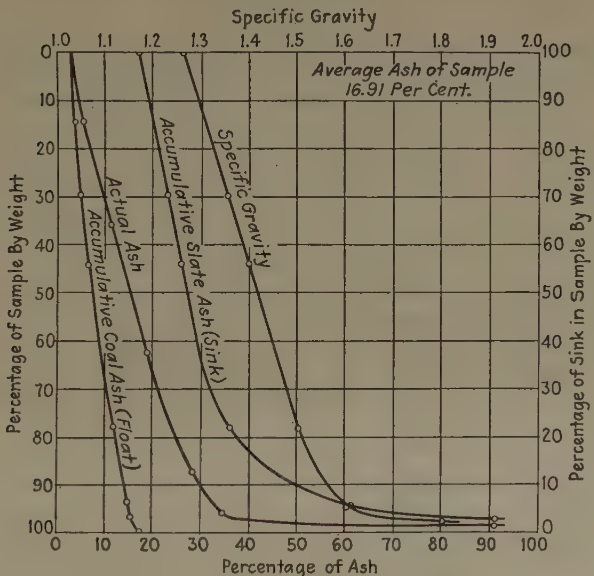

Fig. 1.—Washability curve for 4 by $2\frac{1}{2}$-in. egg, Pocahontas coal.

Note that definite break occurs at about 1.60 sp. gr., and washing at this gravity removes all heavy rock and slate, amounting to about 5 per cent.

Curve 2 indicates a bony coal, and washed egg coal might not have a good appearance, and would be condemned by visual inspection, although it would be well prepared mechanically.

to attain the greatest realization for the operator and yet produce a market product acceptable to the consumer. Preparation is not new in the southern field, but some new angles have presented themselves in recent years on account of the progress of mechanical preparation. Mechanical methods are entirely devoid of the five senses given to

* Koppers-Rheolaveur Co.

human beings and these mechanical devices will always act and behave like machines.

What the writer has in mind most of all is the fact that the measure of the performance of mechanical methods must be made by more scientific methods than rule of thumb or "visual inspection," which has such a stranglehold on the anthracite industry at the present time that it is difficult for machines to function under the accepted standards.

To analyze the problem of the Pocahontas operator and reach some logical conclusions, the writer presents a typical résumé of a washability study of the coals 3 and 4 (Fig. 1). It is a general type, but it must be understood that the concentrations of the impurities in the egg, nut and pea sizes may vary in the different local fields. In certain districts of the Pocahontas field, like the North Fork and Crane Creek areas, the ½ by 0-in. coal is higher in ash than is shown here.

TABLE 1.—*Theoretical Ash Set-ups for Pocahontas Coals 3 and 4*

Size	Pocahontas No. 3			Pocahontas No. 4		
	Recovery, Per Cent.	Ash in Washed Coal, Per Cent.	Ash in Refuse, Per Cent.	Recovery, Per Cent.	Ash in Washed Coal, Per Cent.	Ash in Refuse, Per Cent.
		At 1.45 Sp. Gr.			At 1.40 Sp. Gr.	
Egg	79.60	5.0	43.4	56.5	5.6	39.0
Nut	86.35	5.3	46.7	77.5	5.3	39.0
Pea	90.90	4.2	42.5	85.3	4.9	36.0
Total 5 × ½ in.	86.07	4.8	44.5	74.5	5.0	37.5
		At 1.60 Sp. Gr.			At 1.46 Sp. Gr.	
Egg	85.67	6.6	55.4	81.7	5.7	45.7
Nut	90.95	6.2	58.4	83.0	5.8	46.0
Pea	95.39	5.1	60.1	90.0	5.2	46.9
Total in 5 × ½ in.	91.01	5.9	55.4	81.7	5.7	45.7
		At 1.80 Sp. Gr.			At 1.55 Sp. Gr.	
Egg	93.0	9.0	65.0	75.0	9.3	51.5
Nut	94.0	7.3	70.0	87.1	7.3	55.0
Pea	96.5	5.5	68.0	92.1	5.7	53.0
Total 5 × ½ in.	94.0	6.9	66.5	86.2	7.0	53.0
		At 2.00 Sp. Gr.			At 1.65 Sp. Gr.	
Egg	97.06	10.7	82.9	81.8	11.5	57.0
Nut	96.91	8.7	83.0	90.5	8.0	60.0
Pea	97.63	5.8	80.8	94.0	6.1	62.0
Total 5 × ½ in.	97.1	8.3	82.5	90.0	8.0	59.0

Some rational observations may be made on the set-ups in Table 1, showing theoretical considerations:

1. The distinction in quality between the No. 3 and No. 4 beds of coal is shown. The latter is much the dirtier of the two.

TABLE 2.—*A Three-product Separation of Pocahontas No. 3 Coal*

Size	Size in Feed, Per Cent.	Size, Per Cent. by Wt.	Feed, Per Cent. by Wt.	Ash, Per Cent.	Size in Feed, Per Cent.	Size, Per Cent. by Wt.	Feed, Per Cent. by Wt.	Ash, Per Cent. by Wt.
WASHED COAL, FLOAT, 1.45								
Egg	12.65	79.60	10.07	5.0				
Nut	21.53	86.35	18.60	5.3				
Pea	15.80	90.90	14.37	4.2				
½ × 0 in.	50.02	100.00	50.02	5.2				
5 × 0 in.	100.00	93.06	93.06	5.0				
BONE								
	Float 1.85, Sink 1.45				Float 1.65, Sink 1.45			
Egg	12.65	15.00	1.90	33.24	12.65	7.8	0.99	26.8
Nut	21.53	8.25	1.77	30.50	21.53	5.35	1.15	25.8
Pea	15.80	6.10	0.96	26.10	15.80	4.90	0.77	24.0
5 × ½ in.	49.98	9.26	4.63	30.70	49.98	5.80	2.91	25.7
ROCK								
	Sink 1.85				Sink 1.65			
Egg	12.65	5.4	0.68	71.0	12.65	12.60	1.59	53.5
Nut	21.53	5.4	1.16	72.5	21.53	8.30	1.78	61.0
Pea	15.80	3.0	0.47	73.0	15.80	4.20	0.66	62.0
5 × ½ in.	49.98	4.63	2.31	72.0	49.98	8.09	4.03	58.2

2. Four possible theoretical ash set-ups are shown for each of the two beds under consideration. These set-ups are such that the composite 5 by ½ in. will contain respectively about 5 per cent., 6 per cent., 7 per cent. and 8 per cent. of ash. The specific ash carried by the egg, stove and pea sizes, and the attendant recovery of each, are also given.

3. The logical conclusion from the theoretical set-ups is that, on account of the preparation loss, it is not economical on a two-product separation to clean mechanically either the No. 3 or No. 4 beds much below 1.60 or 1.55 washing gravity. Even then, on bed 3 the preparation loss on egg size at 1.60 sp. gr. is 14.33 per cent. and on bed 4 at 1.55 sp. gr. it is 25.0 per cent. The practical ash in the first case must be about 7.0 per cent. on egg and in the second case, about 9.7 per cent. The washability curves, particularly No. 2, show the difficulties involved.

The main point in the foregoing discussion is that when making a two-product separation the Pocahontas operator should make that separation at a rational point where only the heavy-gravity material is discarded by the cleaning equipment. He should go further; he should

TABLE 3.—*A Three-product Separation of Pocahontas No. 4 Coal*

Size	Size in, Feed, Per Cent.	Size, Per Cent. by Wt.	Feed, Per Cent. by Wt.	Ash, Per Cent.	Size Feed, Per Cent.	Size, Per Cent. by Wt.	Feed, Per Cent. by Wt.	Ash, Per Cent. by Wt.
WASHED COAL, FLOAT, 1.45								
Egg	10.50	65.7	6.90	6.6				
Nut	23.26	83.0	19.30	5.8				
Pea	16.55	90.0	14.90	5.2				
½ × 0 in.	49.69	100.0	49.69	6.0				
5 × 0	100.00	90.79	90.79	5.9				
BONE								
	Float 1.80, Sink 1.46				Float 1.60, Sink 1.46			
Egg	10.50	25.56	2.68	36.3	10.50	13.16	1.38	30.0
Nut	23.26	11.35	2.64	33.9	23.26	5.91	1.41	27.2
Pea	16.55	7.84	1.30	36.6	16.55	3.40	0.57	24.9
5 × ½ inch	49.69	13.32	6.62	35.3	49.69	6.75	3.36	27.9
ROCK								
	Sink 1.80				Sink 1.60			
Egg	10.50	8.74	0.92	69.6	10.50	21.14	2.22	54.0
Nut	23.26	5.65	1.31	69.6	23.26	11.09	2.54	56.1
Pea	16.55	2.16	0.36	81.0	16.55	6.60	1.09	57.4
5 × ½ inch	49.69	5.15	2.59	71.2	49.69	11.72	5.85	55.6

base his market product entirely on such a standard and not be influenced by visual inspection or the sales organization, which, if left to its own resources, may get him into such an economic jam that he will have nothing much to sell in the top prepared sizes, or make the sales price prohibitive in a highly competitive market.

The problem is still more complicated when, on a buyer's market, the operator is mining and mechanically cleaning beds 3 and 4 together. Because of the difference in the character of the two coals and the difference in the normal washing gravities, the market product may be condemned by visual inspection on account of some pieces of bad appearance in the egg size coming from the No. 4 bed of coal.

In any event, this visual inspection is bad business from a technical standpoint and the sooner the Pocahontas coal operator sets up a definite standard of tolerance based on scientific data, the better off he is going to be in the future. J. B. Morrow treated this subject carefully in his paper before the Rocky Mountain Coal Mining Institute held at Denver,

Colorado, May 26–27–28, 1930 (Preparation of Coal for Domestic Market), and the present writer will give his ideas on the subject more fully under a separate head.

Three-product Separation

The problem of the proper mechanical preparation of Pocahontas coals may be studied still further by a three-product separation in which it is proposed to use the intermediate, or bone product, for low-grade fuel purposes, as it is common knowledge that the bone is the greatest cause of trouble in mechanical preparation. These studies are set forth in Tables 2 and 3.

Discussion of Three-product Separation

From a study of the three-product separation tables, the following observations may be made:

1. It would appear that a type three-product separation may be made at about the following gravity cuts: 1.45 sp. gr., low-ash product; 1.45 to 1.60 sp. gr., bone or boiler coal; 1.60 sp. gr., refuse or rock.

2. The recoveries and ash in the various products on the basis of these separations are approximately as shown in Table 4.

Table 4.—*Recoveries and Ash in Three-product Separation*

Product and Size, In.	Pocahontas No. 3		Pocahontas No. 4	
	Recovery, Per Cent.	Ash, Per Cent.	Recovery, Per Cent.	Ash, Per Cent.
Low ash (5 × 0)	93.06	5.0	90.79	5.9
Bone (5 × ½)	2.91	25.7	3.36	27.9
Rock (5 × ½)	4.03	58.2	5.85	55.6
Total and average	100.00	7.8	100.00	9.5

This seems to be a sensible method of handling the bone grief in the Pocahontas coals where the bone can be used for firing boilers equipped with mechanical stokers for burning a high-ash material. This method gives a low-ash market product, probably about 5. 5 per cent. in the No. 3 and 6.5 per cent. in the No. 4 composite, and the egg size free from the bone that spoils its appearance in the railroad cars. Furthermore, only real rock is sent to the dump, making the net recovery high.

Preparation Loss and Degradation

To make a definite comparison of the theoretical with practical results of mechanical preparation, Table 5 is set up. This table deals with "preparation loss" caused by cleaning and degradation due to handling from the pit car through the tipple, cleaning plant and

TABLE 5.—*Preparation Loss and Degradation of Pocahontas Coals 3 and 4*

	Size, In.	Raw Coal, Per Cent. (a)	Raw Coal, Per Cent. (b)	Washed Coal, Per Cent. (a)	Washed Coal, Per Cent. (b)	Washed Coal on Shipped Product Basis, Per Cent.
No. 3 Seam	Lump (plus 5)	......	15.00	.....	15.00	15.93
	Egg (2 × 2½)	12.65	10.70	10.07	8.55	9.07
	Stove (2½ × 1)	21.53	18.27	18.60	15.80	16.77
	Pea (1 × ½)	15.80	13.42	14.37	12.20	12.95
	Slack (½ × 0)	50.02	42.61	50.02	42.61	45.28
	Total	100.00	100.00	93.06	94.16	100.00
No. 4 Seam	Lump (plus 5)	......	15.00	.....	15.00	16.28
	(2 × 2½)	10.50	8.94	6.90	5.86	6.36
	Stove (2½ × 1)	23.26	19.76	19.30	16.40	17.78
	Pea (1 × ½)	16.55	14.08	14.90	12.66	13.73
	Slack (½ × 0)	49.69	42.22	49.69	42.20	45.85
	Total	100.00	100.00	90.79	92.12	100.00

		Theoretical Washed Coal (After Dump of Pit Cars), Per Cent. No. 3	No. 4	Combined	Shipped Product Railroad Weights, Per Cent.	Preparation Loss and Degradation, Per Cent.
Coals 3 and 4 Combined	Lump (plus 5)	15.93	16.28	16.11	13.0	−3.11
	Egg (5 × 2½)	9.07	6.36	7.72	9.0	+1.28
	Stove (2½ × 1)	16.77	17.78	17.28	13.0	−4.28
	Pea (1 × ½)	12.95	13.73	13.34	10.0	−3.34
	Slack (½ × 0)	45.28	45.85	45.55	55.0	+9.45
	Total	100.00	100.00	100.00	100.0	0.00

[a] Percentages in terms of 5 by 0-in. coal.
[b] Percentages in terms of run of mine coal.

WASHERY PERFORMANCE (2-PRODUCT SEPARATION)
(Theoretical @ 1.60 sp. gr.)

Size, In.	Ratio of Mining, One Part No. 3–One Part No. 4: Washed Coal, Ash Per Cent.	Recovery, Per Cent.
Egg (5 × 2½)	8.5	82.2
Stove (2½ × 1)	6.7	89.9
Pea (1 × ½)	5.5	94.4
Slack (½ × 0)	5.6	100.0
Average	6.1	94.8

Final reject on 85 per cent. of run of mine = 5.2 per cent.; on 100 per cent of run of mine, 4.42 per cent.

screening station. The résumé tells the story: 5-in. lump, −3.11 per cent.; 5 by 2½-in. egg, +1.28; 2½ by 1-in. stove, −4.28; 1 by ½-in. pea, −3.38; ½ by 0-in. slack, +9.45. It shows the behavior of soft friable coals like the Pocahontas coals in a tipple and in a complete preparation plant, where every effort has been made to "handle the coal like eggs." The lump has lost over 3 per cent. but the egg has picked up about 1.25 per cent. Considering the "prepared sizes" as +½-in. coal there has been a total loss of 9.45 per cent., of which 4.42 per cent. is chargeable to preparation loss and 5.02 per cent. to degradation through the plant.[1]

In passing, it may be remarked that in anthracite practice, under most favorable conditions, the total preparation loss, including roll reduction and degradation, may easily be 25.0 per cent., so that the Pocahontas operator has something to be thankful for after all.

TABLE 6.—*Theoretical and Practical Washing at Two Gravities*

Size, In.	Pocahontas No. 3, Three Parts			Pocahontas No. 4, One Part		
	Raw Coal, Ash Per Cent.	Theoretical Washed Coal, Ash Per Cent.		Raw Coal, Ash Per Cent.	Theoretical Washed Coal, Ash Per Cent.	
		1.60 Sp. Gr.	1.65 Sp. Gr.		1.60 Sp. Gr.	1.65 Sp. Gr.
Egg (5 × 2½)	12.8	6.6	7.1	19.7	10.5	11.5
Stove (2½ × 1)	11.0	6.2	6.8	12.6	7.2	8.0
Pea (1 × ½)	7.6	5.1	5.2	9.3	5.9	6.0
Slack (½ × 0)	5.2	5.2		6.0	6.0	
Average	7.8	5.9 (5″ × ½″)	6.3	9.5	7.4 (5″ × ½″)	7.9

ON THE RATIO OF MINING

	Raw Coal, Ash, Per Cent.	Theoretical Washed Coal, Ash Per Cent.		Market Product, Ash, Per Cent. Avg. 1.65 Sp. Gr.
		1.60 Sp. Gr.	1.65 Sp. Gr.	
Egg (5 × 2½)	14.5	7.6 (5 × 2¾)	8.2	8.6
Stove (2½ × 1)	11.4	6.5 (2¾ × 1¼)	7.1	6.9
Pea (1 × ½)	8.0	5.3 (1¼ × ⅜)	5.4	6.1
Average		6.4	6.7	6.9
Refuse, ash, per cent.				68.2

[1] In Table 5 the preparation loss has been figured out of the picture, and the 9.45 per cent. referred to is actually total degradation. A certain part of this degradation is recovered as a small-sized shipped product, and full credit should be taken for it, as it is shipped separately. In this particular case it happens to be about 5 per cent., so that the actual degradation into slack sizes, ½ by 0 in., would be 4.45 per cent.

Table 7.—*Details of Washery Performance*

Size, In.	Raw Coal Three Parts No. 3 and One Part No. 4		Washed Coal Theoretical @ 1.60 Sp. Gr. and 1.65 Sp. Gr.		
	Percentage by Weight, Per Cent.	Ash, Per Cent.	Percentage by Weight,	Ash, Per Cent.	Ash, Per Cent.
Egg (5 × 2½).........	10.26	14.5	8.40	7.6	8.2
Stove (2½ × 1)........	18.64	11.4	17.02	6.5	7.1
Pea (1 × ½)...........	13.58	8.0	13.14	5.3	5.4
Total and average....	42.48	11.6	38.56	6.4	6.7

Sink-and-float Data and Chemical Analysis

Sample	Egg Coal, 9 Per Cent. 1.60 to 1.65 Sp. Gr.			Stove Coal, 13 Per Cent. 1.60 to 1.65 Sp. Gr.			Pea Coal, 18 Per Cent. 1.60 to 1.65 Sp. Gr.			Refuse, 1.60 Sp. Gr.	
	Sink, Per Cent.		Ash, Per Cent.	Sink, Per Cent.		Ash, Per Cent.	Sink, Per Cent.		Ash, Per Cent.	Float, Per Cent.	Refuse, Ash Per Cent.
No. 1			7.4			6.3			6.0		
2			10.4			6.5			6.0		66.7
3	5.6	1.0	9.4	2.8	0.2	7.0	1.2	0.6	6.1	2.7	69.5
4	4.7	1.3	10.1	1.8	0.1	7.5	1.2	0.6	6.2	3.5	72.6
5	3.3	0.6	8.1	4.7	0.8	7.3	2.0	0.5	6.3	3.0	60.1
6	5.3	1.0	9.9	3.6	1.1	8.0	1.6	0.8	7.0	0.9	71.2
7	3.6	1.0	9.3	2.6	0.8	6.5	1.4	0.7	6.2	1.9	69.3
8	3.5	2.1	10.1	1.3	0.8	8.3	0.9	1.1	6.3	Refuse, 1.65 Sp. Gr.	
9		0.0	6.9		1.6	7.0		0.8	6.0		
10		0.5			1.0			0.8		6.1	
11		2.2	7.7		1.8	6.6		1.0	5.7	5.0	
12		3.3	9.8		1.4	6.9		1.3	6.3	2.7	
13		0.0	7.3		1.1	6.7		0.8	6.1	2.6	
14		2.0	9.1		0.8	6.4		0.6	6.0	6.0	
15		0.0	8.0		0.6	6.0		0.5	6.2	6.7	
16		0.0	7.6		0.5	6.2		0.6	5.8	2.7	
17		1.3	8.2		1.0	7.1		0.6	5.8	1.0	
	26.00	16.50	138.30	16.80	13.60	110.30	8.30	11.30	98.00	32.80	409.40
Average.......	4.33	1.1	8.64	2.80	0.91	6.89	1.39	0.75	6.12	4.1	68.23

Average Sink, Per Cent.

Average sink all sizes (5 in. × ½ in.) @ 1.65 sp. gr. =......... 0.88
Average sink all sizes raw coal @ 1.65 sp. gr. =................ 8.50

Efficiencies @ 1.65 Sp. Gr., Per Cent.

Qualitative.. 90.6
Quantitative... 99.7
Bank loss.. 0.3

Average Ash, Per Cent.

Raw coal (5 in. × ½ in.)................................. 11.6
Washed coal (5 in. × ½ in.).............................. 6.9
Theoretical washed coal (5 in. × ½ in.).................. 6.7

Washing at Two Gravities

Table 6 gives a comparison between theoretical and practical washing at two different gravities on the two Pocahontas coals when washed together. From a chemical analysis standpoint the practical washing is eminently satisfactory but from a visual inspection standpoint, the egg might not be good on account of its bony appearance. The point is illustrated as follows:

Size	Theoretical Washing, Ash Per Cent.	Practical Washing, Ash Per Cent.
Egg	8.2	8.6
Stove	7.1	6.9
Pea	5.4	6.1
Average	6.7	6.9

It can hardly be expected that mechanical preparation could do any better than the performance described.

Washery Performance

The details of washery performance are shown in Tables 7 and 8.

Table 8.—*Three-product Separation*

Coal, 1.45 Sp. Gr., Bone, 1.45 to 1.60 Sp. Gr., Rock, 1.60 Sp. Gr.

Sample No.	Ash, Per Cent.							
	Egg		Stove		Nut		Sludge	⅛-in Dry. Slack'
	Raw	Washed	Raw	Washed	Raw	Washed		
1	7.4	5.4	10.2	6.7	7.4	5.3	4.3	5.1
2	7.2	5.0	10.6	6.2	6.4	5.7	4.4	5.2
3	6.4	4.8	10.1	7.1	7.4	5.5	4.3	5.7
4	13.0	5.7	13.7	6.4	7.7	4.9	4.1	5.5
5	10.6	5.3	13.4	6.3	9.9	5.8	4.1	5.9
6	10.4	4.7	9.9	6.1	7.0	5.4	4.2	5.9
7	10.2	5.8	10.1	6.7	8.0	4.9	4.2	6.1
8	9.4	6.8	13.0	5.7	8.6	4.9	4.4	5.0
9	8.9		13.5		9.5		4.2	5.1
Average	9.3	5.4	11.6	6.4	7.9	5.3	4.2	5.5

Perhaps the three-product system is the ideal way in which to handle Pocahontas coals where the bone can be used as a boiler fuel. In the operation detailed in Tables 7 and 8 the boiler coal is held at about 30 per cent. ash, which is said to be entirely satisfactory when fired in a moist condition. The washed coal is practically free from heavy-gravity material; the final refuse at 1.60 sp. gr. is reasonably free from

float material and at 1.40 sp. gr. the float is fairly low, which indicates good washery performance.

Tentative Definition for Clean Coal and Tolerance for Impurities

The writer submits a tentative definition for clean coal and the tolerances for impurities in the different size of Pocahontas coal, as follows:

Clean coal may be defined as the product of a preparation plant that shall not contain more than 2 per cent. of intermediate ash sink in all sizes cleaned, as determined by sink and float methods at whatever washing gravity is set up from the washability study.

The tolerances for the impurities in the market products may be:

Lump	Hand-picked completely
Egg (5 × 2¾)	Under 1 per cent. sink
Stove (2¾ × 2)	1 to 2 per cent.
Nut (2 × 1⅛)	1 to 2 per cent.
Pea (1⅛ × ½)	2 to 3 per cent.
Slack (½ × 0)	3 to 4 per cent.

Practically, clean coal is coal that is free from the extraneous impurities of rock, slate, pyrite, etc. and the percentage of ash content is not the governing factor, as the inherent ash varies widely in different coals. The tolerance for actual slate and rock may be set as 25 per cent. of the above figures and the defining gravity may be set at about 1.80 specific gravity.

The foregoing statements indicate that a 50-ton car of Pocahontas egg coal under favorable conditions of mechanical preparation may contain 1000 lb. of heavy-gravity material of which 250 lb. may be rock or slate. Stove and nut coal may observe the same restrictions or doubled, depending upon the conditions.

The writer invites a full discussion of these standards for Pocahontas coal as, in his opinion, the future of mechanical preparation in the field, and the very life of the coal industry itself, is dependent upon the adoption of a fair and reasonable standard for the market product based on scientific methods—not rule of thumb.

DISCUSSION

(*Chester M. Lingle presiding*)

T. W. Guy, Charleston, W. Va.—In connection with the proposed standard of preparation for Pocahontas coal, a number of operators have expressed themselves as appreciating the importance of establishing some such standard for the preparation of that coal. It should be done in all coal fields, because to a very large extent the sales people and officials who determine the sales policy of the coal companies are prone to market their coal without giving enough consideration to whether it can be

prepared economically to meet specifications, which are constantly getting more difficult to meet in a continuing buyer's market.

In the Pocahontas field, with the lump, egg and other prepared sizes, the fact that makes it valuable is that it brings a high price for domestic use. There you have to depend upon what the customer, the ultimate consumer, gets from his visual inspection. To him, ash or specific gravity means little, but a few pieces with a laminated appearance, or a thin streak of slate, bone or pyrite on a few pieces of coal, will cause a serious complaint, even though the pieces in question may have only 8 or 10 per cent. ash and may float at 1.40° gr. Mr. Campbell properly emphasizes the economic waste and the heavy burden imposed upon the operator in allowing such unreasonable standards to become established in his markets.

The difficulty of the Pocahontas operators in arriving at definite standards is greatly complicated by the fact that a few mines have much better conditions than the others. They do not have streaks of bone or pyrite or laminated material to such an extent in their prepared coal. The result is that the standard of coal they would set, or attempt to set, is much higher than the average mine in the field or than those that are less favorably situated can meet. That is true to a large extent in the other fields as well.

F. A. Jordan, Youngstown, Ohio.—Is there any run of mine Pocahontas coal that will meet these tolerances?

J. R. Campbell.—I think not, Mr. Jordan.

F. A. Jordan.—That means it will all have to be cleaned.

J. R. Campbell.—None that I have seen would meet those tolerances, mine run. Perhaps Mr. Guy can answer that better than I. There are some very clean Pocahontas coals, I understand.

T. W. Guy.—I referred to prepared sizes. Some mines claim to be shipping 6 and 7 per cent. mine run, and I think there are some mines whose average is close to that. That, however, indicates that there the fines have low ash. In some mines of the Pocahontas field, the fines run only around 5 per cent. ash normally, and since the fines are the greater part of the coal, naturally they get low ash in the mine run. In other mines in the Pocahontas field, the fines through ½ in. will go 10 or 12 per cent. or higher. Since they are 50 to 70 per cent. of the coal, naturally the ash in the mine run is high.

The lump, or rather the middle sizes, are almost universally high in ash before they are cleaned, either by hand-picking or mechanically. At the best mines the prepared coal will come within the proposed tolerances as to gravity, and also be practically free of the pieces of lower gravity which are objectionable because of appearance.

R. E. Rightmire, Fairmont, W. Va.—I do not believe it is practicable to have a standard of a tolerance of 1 per cent. sink in Pocahontas mine-run coal as marketed unless we have such tests on the higher gravities. Of the different gravities that the author of the paper has considered, I think that generally 1.60° is a favorable gravity for the mechanical cleaning of Pocahontas coal. I would consider that 1 per cent. tolerance in sinks in cleaned coal at that gravity would not be desirable for the producer, as that basis would draw the lines too tight on practical cleaning plant operation.

J. R. Campbell.—It is only something to shoot at.

R. E. Rightmire.—Well, taking a little shot, if the gravity basis were 1.80 instead of 1.60, I would say that 1 per cent. tolerance should be fairly easy, but at the 1.80°

gravity we probably would not have the quality of Pocahontas we would like to have. In other words, there would be too much of the twilight material in the cleaned product and there would be some of the heavier bands and binders as well.

Another thought brought out by Mr. Campbell in the early part of his comment on mechanical methods is their freedom from the five senses of human beings. I think this comment is very timely, for it has been my observation that frequently we find the attempt to measure the standard of mechanically cleaned coal by visual inspection. A mechanical cleaning plant of whatever process does not know anything at all about appearance of coal—all it knows is a gravity distinction and it must operate on that basis. The author brings out another important consideration in connection with the marketing feature, when he says that the quality of mechanical cleaning should not be passed upon, or should not be measured by visual inspection. In this, I think he is absolutely right, for frequently we find material in cleaned coal that has a very inferior appearance, which absolutely conforms to the float-sink separation, yet from the standpoint of visual inspection we would throw it out. It would be unfair to condemn any cleaning process or cleaned product for including such material, because it is found where it rightfully belongs.

In the exhibits that Mr. Campbell has given, I was a little surprised to notice the analytical showing of the Pocahontas No. 4 sample and I wonder, in that connection, whether the No. 3 and the No. 4 samples are supposed to be representative of the best of their class or whether they are just samples taken at random. We know there are some comparatively poor Pocahontas No. 4 coals and there are also the exceptions which show very good quality. Undoubtedly it is also true there are some Pocahontas No. 3 coals which show considerably poorer quality than the samples given in the exhibit.

The question of tolerances as brought forth by the author is a difficult one. The proposed tolerance for egg coal appears to be tightly drawn and while some of the other proposals are more liberal they are drawn tight enough to cover closely the average situation of production and preparation. There are many irregularities and inequalities to be adjusted but on the broad question of preparation and marketing of Pocahontas coals the author's comments and suggestions are well worth consideration as a solution.

T. A. Stroup, Omar, W. Va. (written discussion).—Mr. Campbell's paper brings out a thought that is of great interest to everyone who contemplates treating large sizes of bituminous coal; that is, that three products usually result from a washing process that will give a clean coal and a clean refuse. This middle-gravity material, which may be bone or may be coal with adhering pieces of slate, must be disposed of in some way. When one is treating slack this middle product can be recirculated until the individual pieces either go into the clean coal or into the slate, but such a course is not practical with large sizes of coal. The company treating large coal which has a steam plant in which this middling product can be burned is indeed fortunate. Those who cannot dispose of this troublesome material in this way are forced either to waste it, with a resulting great waste of coal, or to crush it to slack size and send to the slack washer, which results in lower realization.

The fact that so many egg-coal washers are guilty of losing a great deal of coal with the refuse is due largely to the fact that the importance of a middling product in making a good job of cleaning this size of coal has been overlooked. On the other hand, attempts to reduce the loss of coal in the refuse lead to more dirt in the cleaned product.

Unless a coal is exceptionally free from middle-gravity material no attempt should be made to wash it without an adequate study of the disposition of this intermediate product.

R. W. Arms, Chicago, Ill.—I have no doubt that the operators in West Virginia would appreciate it highly if Mr. Campbell could make this thing stick. It would be very desirable if he could establish a basis on which they could ship their coal and know that it was all right when it left the mine. Unfortunately, however, in the final analysis the domestic consumer is the judge of Pocahontas coal, and the domestic consumer can find more things to complain about in a minute than any other type of consumer. On pea coal and nut coal there have been at least a dozen reasons for turning down a carload of coal, even though they might have lived up to these tolerances. I have seen both dry-cleaned coal and wet-cleaned coal rejected, not for any reason connected with cleaning, but simply because of the amount of fines and the degradation, or lack of screening in the beginning, or for many other reasons.

If we could, by a long campaign, instruct the domestic coal-consuming public that this list of tolerances was the right one, we certainly would be accomplishing something, and something important. The operators would then accept any list of tolerances that we might set up and live up to them. There is no extreme technical difficulty in living up to this list of tolerances, with the possible exception of egg coal in those seams, but to live up to the domestic consumer's demands as they have existed in the past is a tremendously difficult thing.

Conditioning of Coal for Treatment by Pneumatic Cleaners

By Thomas Fraser,* Hazleton, Pa. and Robert MacLachlan,† Library, Pa.

(Pittsburgh Meeting, September, 1930)

The dry cleaning of coal is a relatively new art and, as might have been expected, a number of unforeseen problems have been brought to light in the few years that dry methods of treatment have been in commercial use. The pneumatic concentrators that were first adapted to the cleaning of coal had already been developed to a fair degree of efficiency, although at low ratings, in the field of ore dressing. The concentrating machines, therefore, were in a relatively more perfected state than the general technology of dry cleaning and, for this reason, the major difficulties encountered in the early plants were in the accessory operations involved in the servicing of the cleaners rather than in the functioning of the cleaners themselves. The crudities of a new technology were further aggravated, in some cases, by meagerness of appropriations which precluded even the provision of adequate storage bins and such other facilities as had been long recognized as indispensable to the proper functioning of older types of cleaning plants.

The practice of dry cleaning is now sufficiently established so that we may examine critically auxiliary equipment and operations. The cleaning processes have been so simplified, with respect to presizing of the raw coal required and the quantity of air circulated, that stinting of expenditures for auxiliary equipment is not necessary to keep the per ton cost of installation in line with complete and well designed plants of other types. Furthermore, sufficient operating experience has been accumulated to ascertain the conditions most conducive to effective operation of dry-cleaning machines; at least the essentials of a definite technology of dry cleaning has been established.

Early Operating Problems

The problems that have occupied the builders and operators of dry-cleaning plants from the beginning have been relatively few, but difficult of solution. The principal difficulties have been:

1. The proper sizing of the raw coal to suit the cleaning process.
2. The handling of raw coal when it is delivered wet from the mine.
3. Dust collecting.
4. The maintenance of uniformity in cleaning performance.

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Sizing of the raw coal, when dry, has become a fairly simple mechanical operation, since the introduction of the modern concentrating machines capable of handling a wider range of sizes, corresponding generally to the schedule of market grades. However, the cleaning of extremely fine sizes still involves some screening difficulties even when the coal is dry. The practice of treating unsized feed[1] on stratifying tables, while eliminating altogether the necessity for presizing of the feed, has introduced a new condition calling for a certain degree of uniformity in size composition. This may also be considered a matter of size control and involves some conditioning of the raw coal before treatment to get the best cleaning effect.

Most of the trouble in screening is experienced, of course, through trying to handle wet coal. Excessive moisture may also occasion some irregularity in the operation of pneumatic tables, particularly if the condition is fluctuating; although the cleaning operation is not affected, as a rule, until the moisture exceeds considerably the percentage that causes trouble in fine screening. In either wet or dry processes of coal cleaning, water, in one way or another, accounts for a very large part of the operating trouble, and if the coal can be delivered to the plant in a uniformly dry condition, a dry plant should provide a simple answer to the cleaning question.

Control of the dust in handling dry coal had already received some attention in tipples and industrial coal-handling plants before dry cleaning came into extensive use. The hazard is considerably increased by the large quantities of air blown through the coal by pneumatic cleaners, and the problem is correspondingly more acute in cleaning plants. With the progressive simplification of the cleaning operation, a marked reduction in the amount of air used and in the mechanical handling of the coal has lightened the dust burden as compared with the early plants. Dust trouble has varied greatly at individual plants. A very slight difference in moisture content will make a great difference in the tendency of the coal to produce dust clouds. Furthermore, the amount of air-floated dust produced in handling varies greatly with different coals; some friable coals, for example, appear to break with the formation of a large proportion of very fine yet granular material which does not rise in the air appreciably, whereas another coal may produce a large proportion of impalpable dust. This appears to be a definite coal characteristic entirely apart from friability in the usual sense.

After being ensured a properly conditioned feed, uniform performance of the cleaning plant depends in a large measure upon the mechanical equipment available to service the cleaners; to supply an uninterrupted

[1] An unsized feed is considered to be any natural mixture of sizes ranging from a definite maximum, such as ½ in., 1 in. or 4 in. down to 0 in. without any minimum size limit.

uniform flow of raw coal, air and power, and to maintain the proper conditions throughout the operating period.

Maintenance of favorable operating conditions without interruption is necessary to the production of a uniform product which is fully as important as purity of product. Buyers are prone to complain of coal deliveries that vary in quality from car to car even through averaging low in impurities; or of occasional shipments that vary widely from specification or from the seller's representations.

Sizing

For sizing dry small coal preparatory to air cleaning, practice is virtually standardized on high-speed vibrating screens. The high-frequency stroke of small amplitude has established its superiority for capacity and freedom from blinding when handling fine material. Vibrating screens of the inclined, gravity-feed type or of the horizontal, conveying type are about equally effective for this service. The gravity-feed type has the advantages of low power requirement and little vibration imparted to the supporting structure; while the horizontal type will make a more precise separation at the screen aperture size and operates with less loss of head room.

Oblong mesh (Ton-cap or Rek-tang) wire cloth is used as the screening medium for the sizes under ½ in., because of its large percentage of open space as compared with either punched plate or square-mesh cloth. A screen fabric made of steel music wire wound on or welded to heavier cross wires spaced 3½ to 4 in. apart gives promise of even greater freedom from blinding. The mesh of these cloths is designated by the net width of opening and may be obtained in fine sizes such as 1/16, ⅛, and 3/16 in., suitable for the sizing of fine coal. The openings in the 1/16-in. cloth, for example, are 1/16 in. wide by 3½ in. long. This gives a large percentage of open space due to the small number of cross wires. The scouring action of the coal passing over the long unobstructed lengths of longitudinal wires prevents the accumulation of mud or fireclay which sometimes bridges over the openings in fine screens.

In the early air-table plants, where the raw coal was separated into a great number of sizes before treatment, and no storage facilities were provided, much difficulty was experienced in both the screening and the cleaning operations because of great variations in the amount of feed going to individual units as the size composition of the mine-run coal changed. This caused irregular loading of both screen and separator units. Difficulties in screening were overcome by providing ample excess screen area, but irregular operation of the table could not be corrected in this manner. Frequent starting and stopping of machines and operation of some machines with too light loads resulted. Simplification of sizing schedules has greatly reduced this source of trouble, but

considerable irregularity in operation of the tables is still unavoidable with the usual kind of flow sheet shown in Fig. 1, providing virtually no storage of conditioned separator feed.

When raw coal storage is provided to enable the cleaning plant to operate uninterruptedly with irregular delivery of coal from the mine, the main storage bins may be located as shown in Fig. 2, to receive raw unsized plant feed; or the bins may be placed after the screens to store the sized feed in condition for treatment, as shown in Fig. 3. With the arrangement shown in Fig. 2, at a plant equipped with tables that require

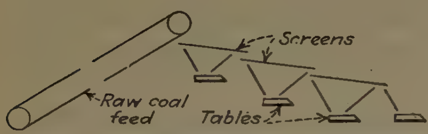

FIG. 1.—USUAL FLOW SHEET OF TABLE PLANTS.

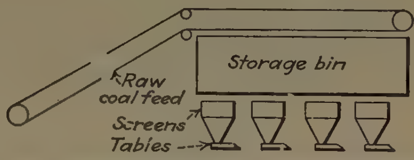

FIG. 2.—TABLE PLANT WITH STORAGE BIN.

a sized feed, the coal is drawn out at a uniform rate to the screens furnishing an ideal feeding arrangement so far as the screening is concerned and enabling the cleaning plant to operate when the mine is idle, so long as there is a supply of coal in the bin. However, this arrangement does not provide a constant feed to the various cleaning units, as each set of machines for any given size has to take all of that product as it is delivered by the screens and the load on any separator will vary as the size composition of the screen feed varies.

The plan shown in Fig. 3 provides for large-scale storage of sized coal in condition for treatment by the separators. Therefore it has the great advantage of providing a uniform uninterrupted feed of sized coal to each separator, regardless of fluctuations in either the rate of production of raw coal or its size composition. It has also the advantage of more compact plant construction than plan 2. When the raw coal is to be treated in three sizes, for example 1¼ by 2½ in; 5/16 by 1¼ in., and 0 by 5/16 in., the raw coal, delivered into the plant by an inclined conveyor, may be sized on two rows of double-deck vibrating screens arranged upon opposite sides of the horizontal run of the conveyor. Separate conveyors are required to carry the three sized products of these screens to the three storage bins. Three sets of separators below these bins are arranged to take the sized coal through individual feeders. This provides positive

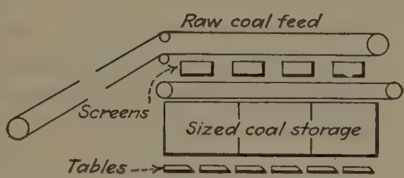

FIG. 3.—LARGE-SCALE STORAGE OF SIZED COAL IN CONDITION FOR TREATMENT BY SEPARATORS.

control of the feed to each cleaning machine, where positive uniformity is most essential. Fluctuations in the feed to screens may be taken care of by installing ample screen capacity to handle the peaks, or by additional storage ahead of the screens.

Storage for raw coal may be provided in the form of a storage bin or in pit cars alloted to the preparation plant. Storage in pit cars at the dump has the advantages of flexibility in handling, reduced tendency to segregation by sizes that often occurs in bins, reduced degradation by handling and better operating conditions for the raw-coal dumping and conveying equipment.

One disadvantage of the plan of storing sized coal, as shown in Fig. 3, is the breakage to which the coal is subjected after it is sized and before it is delivered to the separators. The effect of this breakage on the

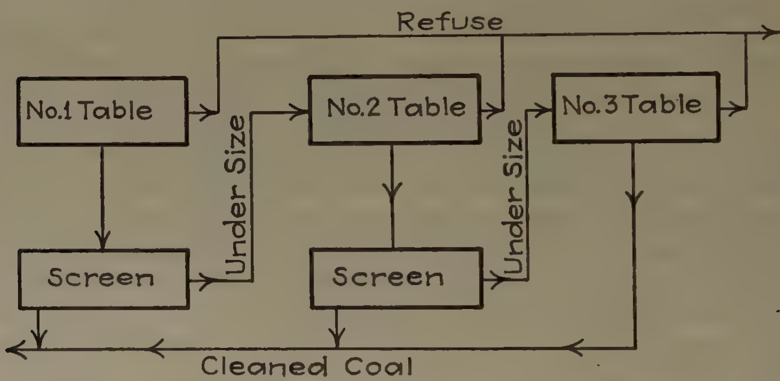

FIG. 4.—FLOW SHEET FOR TREATING RAW COAL UNSIZED AND RE-TREATING FINE PORTION OF CLEANED COAL PRODUCT.

performance of the separators will depend upon the nature of the coal. If the refuse material breaks up into fine particles, these will not be removed in the cleaning treatment and the finished product will be adversely affected. With the more normal condition, where the breakage in handling is virtually all in the coal, no appreciable effect on cleaning performance will result.

With regard to the sizing required to provide a satisfactory feed to pneumatic cleaners, this varies greatly with the nature of the raw coal and the degree of cleaning desired. Experiment has shown that the presence of fine material in the feed to a table increases the range of sizes actually cleaned, even though the extreme fine material itself is not separated. That is to say, when an unsized feed is passed over the machine, the range of sizes actually cleaned is greater than can be handled if this same size range is screened out and fed separately to the table. There is, therefore, a certain advantage in treating the raw coal unsized, and retreating the fine portion of the cleaned coal product that is not

sufficiently cleaned in the primary operation. This plan of operation is shown in Fig. 4. It has the disadvantage of much rehandling and consequent degradation of the recleaned part of the coal, and of increasing the load on the primary coarse coal tables. The advisability of using this arrangement, therefore, depends upon the coal. It would not be suitable for a friable coal containing only a small proportion of the coarse size that would be finished on the primary machines.

In large plants, the average performance when treating a wide range of sizes on the primary tables may be improved by taking a wide middling cut and screening this material to a closer sizing schedule for final cleaning on separate tables. For example, the middling from primary tables handling the ¼-in. to 1¼-in. feed may be separated into two sizes (¾ to 1¼ in. and ¼ to ¾ in.) for separate retreatment.

In a pneumatic separator depending upon stratification of the bed, the use of an unsized feed makes it possible to maintain a mobile condition of the pulp with less air than would be required to treat sized coal. This brings about a nearer approach to a hindered settling condition in the bed and increases the range of sizes handled. The layer of coal on the table deck has an important function, in this case, in regulating and distributing the flow of air. The density of the pulp layer is therefore of great importance and the normal distribution of sizes is essential. The deck resistance may be adjusted to suit various types of coal but a certain amount of fine material in the feed is necessary. Having adjusted other factors to the normal composition of the feed, a measure of uniformity in size composition and moisture content of the feed is necessary to maintain uniform results, otherwise the bed may become too open, or too dense and sluggish, as the conditions fluctuate. Conditioning of the feed in a plant treating the raw coal without prescreening thus implies mixing of the raw coal to assure a feed of uniform composition to the separators.

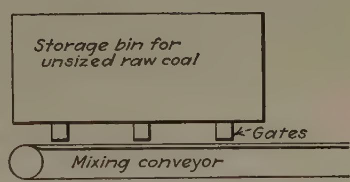

FIG. 5.—STORAGE BIN FOR UNSIZED RAW COAL WITH SEVERAL GATES FOR DELIVERING TO MIXING CONVEYOR.

The use of large storage bins may contribute greatly to this disturbing irregularity in sizes delivered to the cleaners, because of the marked tendency of unsized material to segregate in handling. Complete control of the size composition of the feed could only be obtained by storage of sized coal as shown in Fig. 3 and drawing out known proportions from each bin to a mixing conveyor as fed to the separators. This expedient is more costly than would be resorted to except under unusual circumstances. A sufficient degree of mixing should be obtained by the use of a mixing conveyor receiving coal simultaneously from several gates in the unsized raw-coal storage bin as shown in Fig. 5, or by the storage of

coal in cars which may be dumped on two or more parallel tracks taking coal trips from different sections of the mine. Under this condition, cars may be dumped at a uniform rate to supply the separators with an uninterrupted feed through feed hoppers that are too small for size segregation to take place by gravity flow in the bin.

Thorough blending of the raw coal to obtain a uniform composition of the feed to screens and separators gives the most favorable condition for operation. Uniformity of size composition, moisture content and refuse content, all contribute to uniform functioning of the machines.

Handling Wet Coal

Aside from its role as an undesirable impurity in the product, water in the coal to be treated in a pneumatic cleaning plant causes certain mechanical difficulties, the principals of which are (1) blinding of screens and reduced screening efficiency; (2) blinding of perforations in table decks; (3) increased resistance to air flow through the pulp on a table deck, particularly when fine or unsized feeds are handled.

Varying moisture content is particularly troublesome. Screens once blinded by wet coal are rendered ineffective even for screening dry coal that may subsequently come upon them. A run of wet coal disturbs the operation of a concentrator in which the air and the deck covering have been adjusted to handle dry coal; while a uniformly damp coal, up to a limiting proportion of water, may be handled satisfactorily, if uniform, by adjusting the other factors.

The ideal moisture condition of 0 to ¼-in. coal for treatment on the table is about 3.0 per cent. mechanical moisture content. In the natural mined product where the − ¼-in. size has an average moisture content of 3.0 per cent., the distribution of moisture by sizes below ¼ in. will ordinarily be about as shown in Table 1.

Table 1.—*Typical Distribution of Water*

3.0 Per Cent. Average Moisture Content

Size	Moisture, Per Cent.
¼ in. to 14 mesh	1.5
14 to 28 mesh	4.0
28 to 48 mesh	6.0
48 to 100 mesh	8.0
100 mesh to 0	12.0

The effect of this concentration of moisture in the finest sizes is to increase greatly the screening difficulties if sizing at very fine size, such as 1/16 in., is attempted, but to increase the effectiveness of cleaning of the fines in the unsized feed. The latter effect is probably due to the

increased specific gravity of the moist fine refuse and to the lessened aspirating effect of the dust hoods, which, in treating a bone-dry coal, will take up a considerable part of the clean coal dust and leave slate and clay dust in the table pulp.

Fine screens are more sensitive to the effect of small percentages of moisture than are the concentrators, and fine-coal tables or mixed-feed tables are more subject to fluctuations in resistance of the bed due to moisture changes than are tables fed with sized grain coal.

A vibrating screen clothed with ¼-in. Ton-cap screen (apertures ¼ in. wide by ½ in. long) will operate satisfactorily on coal containing up to 5 per cent. mechanical moisture content. This feed, however, will blind a cloth of ⅛-in. mesh and will give some trouble on 3/16-in. Six per cent. moisture will not cause the ¼-in. cloth to blind but will greatly reduce the effectiveness of the screens. Table 2 gives approximately the effect of moisture content in the feed on the effectiveness of screens. These figures apply to the use of the standard type of oblong mesh cloth on vibrating screens. Sizes refer to net width of opening. Moisture percentages are mechanically held moisture above the natural vein moisture. The data are based upon experience with Pittsburgh coal of about 1.2 per cent. vein moisture. The efficiency figures express the percentage removal of available undersize material present in the unsized coal when the screen is fed at the rate that will give 95 per cent. effectiveness with dry coal—initial condition shown in first line, 2 per cent. moisture.

TABLE 2.—*Effect of Moisture on Efficiency of Screen Fed with Maximum Tonnage Giving* 95 *Per Cent. Screening of Dry Coal*

Moisture, Per Cent.[a]	Removal of Undersize, Per Cent.		
	⅛-in. Screen	3/16-in. Screen	¼-in. Screen
2	95	95	95
3	60	75	85
4	20	50	70
5	Blind	20	40
6		Blind	20

[a] Mechanical moisture percentage = total moisture minus natural vein-moisture content. Size of feed 5/16-in. to 0.

With moist coals, fairly complete screening can be obtained by increasing the screen area per ton of feed in inverse proportion to the above efficiency figures, but the size at which separation is actually made will be smaller than with dry coal on the same screen. As the

moisture approaches the blinding point for any given size (in those parts of table showing efficiencies below 50), close sizing becomes impracticable with any amount of screen area.

Mixing of coal in bins is of little advantage in handling wet and dry coal, because the moisture does not distribute itself through the mixed mass of coal to any great extent; wet lots of coal retain their moisture in segregated parts of the bin.

Predrying of the raw coal, where water in the feed gives trouble in cleaning-plant operation, offers the most positive solution of the difficulty. Operation of a dryer in this service will differ markedly from the drying of wet-washed coal, in that the feed to the dryer will vary in moisture content and a much smaller reduction in moisture content will be required. For these reasons, the regulation of a high-temperature direct heat dryer would be extremely difficult. The small amount of water to be evaporated in most cases would make it feasible to use a dryer at sufficiently low temperature to avoid burning of dry coal. In the usual case, removal of a fraction of one per cent. to 2 or 3 per cent. of moisture will eliminate operating troubles that arise from wet coal in the plant feed. The figures given in Table 3, showing the amount of air at different temperatures theoretically required to carry away 20 lb. of water, 1 per cent. moisture in a ton of coal, give an indication of the air circulation that would be required in such a dryer. These figures are based on 100 per cent. relative humidity in the discharge from the dryer, which it is not practicable to obtain in operation. Hence air consumptions in practice would be some 50 per cent. greater than shown in the table.

Table 3.—*Moisture-removing Capacity of Air-circulation Type of Dryer*
Air Entering Heater at 70° F. and 50 Per Cent. Relative Humidity

Temperature in Dryer, Deg. F.		Weight of Moisture Absorbed per Pound of Air, Grams	Air Required for 20 Lb. Water Removed, Cu. Ft.
Initial	Discharge		
100	68.7	51.2	36,900
120	74.5	74.5	25,400
140	79.6	99.5	19,000
160	84.3	125.0	15,100
180	88.4	151.0	12,500
200	92.3	178.5	10,600

Predrying of the coal has the additional advantage of turning out a uniformly dry final product. With the raw coal perfectly conditioned by drying and subsequent complete screening, the dry preparation plant will produce a cleaned fuel low in ash, sulfur and water content.

SERVICING THE PNEUMATIC SEPARATOR

The various operations of servicing the cleaners include, in addition to drying and sizing the feed, supplying the separators with an uninterrupted feed and disposing of the products; and maintaining favorable working conditions for the attendants.

A most important factor in servicing the preparation plant is a close coordination of mining and preparation activities. Where difficulty is experienced due to wet coal in the feed, much may often be accomplished in the mine to remedy the condition at the source. Wet places may be eliminated by local drainage and a change in haulage schedules may be feasible to avoid wetting of trips left on outside haulage roads at the end of a shift. When wet coal can be eliminated in the mine this is the most economical solution. In any case, a careful study of conditions and methods of working at the face may enable the mine force to produce a raw material more suited to effective treatment in the preparation plant.

This kind of cooperation will sometimes be difficult to establish because production has always had the right of way over all other objectives in the underground organization and any interference with the established routine for any outside purpose may be expected to meet with some opposition. The production of marketable coal is fast becoming a manufacturing process. In this modern scheme of things, the mine ceases to be a complete independent unit producing fuel for the market and becomes a source of raw material for the preparation plant in which the finished market product is manufactured. It will take some time for the mine management to completely recognize this new alignment.

Irregular delivery of coal from the mine is aggravated in most cases by the chronic shortage of cars, which has always necessitated the subservience of all other interests to the paramount object of dumping cars at record speed and getting them back into the mine. This ever-present source of contention between the mine and preparation plant operators could be eliminated by adequate car supply and provision of extra pit cars alloted to the preparation plant in lieu of other storage equipment, to be held when necessary on the storage track at the dumping point.

Uniform uninterrupted feed is essential to the most effective operation of any gravity concentrator depending upon the stratification of the feed pulp in the machine. Retention of a certain amount of bed slate on the separator is necessary to make a good refuse product. A uniform depth and density of moving pulp on the separating surface is also essential to proper regulation of the current whether this be air or water. Some experimental data showing the adverse effect on cleaning performance of an air table caused by stopping and starting of the feed during operation

are given in Table 4. These results were obtained with a table handling 7⁄16 by 1⅛-in. coal containing about 7 per cent. of refuse heavier than 1.6 sp. gr.

TABLE 4.—*Irregularities in Pneumatic Table Performance Caused by Intermittent Feeding*

	Sp. Gr.	Condition I[a]	Condition II[b]	Condition III
Sink in cleaned coal, per cent...	1.6	1.3	7.3	2.0
Float in refuse, per cent........	1.6	9.8	56.0	21.6

[a] With uniform constant feed.
[b] During 1 min. after starting following a stop in the feed.
[c] Average of actual operation including irregular stops.

Other factors of importance in the dry treatment of coal are a well designed and constructed plant to assure the minimum of interruption due to failure of coal-handling equipment; provision of adequate light and space for convenient operation and repair of equipment; general maintenance of plant in clean and orderly condition; manning of the plant with intelligent and well trained attendants; and analytical control of the plant with daily posting of results for the information of plant operators.

The operation of a gravity concentrator is an art which cannot be entirely regulated mechanically, no matter how complete and expensive the equipment; nor can it be entrusted to the care of unskilled or uninterested workmen. No entirely automatic control has yet been devised to work satisfactorily but all concentrating devices give their best results under the care of a long-experienced and interested operator who is able to "sense" certain maladjustments and their effects on performance. In this respect the cleaning plant presents certain operating problems different from other mechanical equipment in more common use at bituminous mines. Recognition of this factor and the introduction of systematic control and information of machine operators has resulted in striking improvement in cleaning performance in some instances. The psychological effect of good light, general cleanliness and plant order is probably of more importance in the cleaning plant than in other branches of mining work.

The most important conditions requisite to uniformly effective pneumatic cleaning-plant performance are:

1. Regular continuous feed of raw coal.
2. Proper sizing of feed to suit the cleaning process.
3. Uniform low water content of raw coal as delivered to cleaners.
4. Uniform size composition of feed to machines taking unsized feed.
5. Adequate and dependable coal-handling equipment.
6. Skilled attendance under favorable working conditions.

DISCUSSION

(Chester M. Lingle presiding)

K. C. APPLEYARD, Birtley, England.—On page 289 the authors deal with the difficulties of screening when wet coal is being used, and suggest that treating of unsized feed eliminates entirely, or almost entirely, the necessity for presizing, but introduces a new condition calling for a certain degree of uniformity in size composition. They go on to say, "Most of the trouble in screening is experienced, of course, through trying to handle wet coal. Excessive moisture may also occasion some irregularity in the operation of pneumatic tables, particularly if the condition is fluctuating."

Our experience in England is that moisture does not worry us at all down to about ⅛ in., and we have wet-screened coal down to ⅛ in. and fed wet-screened coal on to pneumatic separating tables and dry-cleaned it perfectly. That is just in the way of experiment.

On page 295 the authors say that "6 per cent. moisture will not cause the ¼-in. cloth to blind, but will greatly reduce the effectiveness of the screens." I think the question of moisture wants rather more investigation than it has had. In our experience the troubles of moisture are largely due to differences in the refuse. If there is a soft refuse, a clay refuse, and it is in contact with water for any considerable time, it disintegrates, becomes sticky, sticks small particles together and then can be neither screened nor dry-cleaned. On the other hand, if the refuse is hard and does not disintegrate in the presence of moisture, there is, as a rule, not a great deal of difficulty in screening or in cleaning it.

I have seen a screen—dare I say even a ½-in. Hum-mer screen—blind up with a coal containing 5 per cent. of moisture; blind up after some considerable working; simply because the 5 per cent. of moisture had been in contact for some time with a soft, sticky refuse, which stuck the coal particles together so that there was a blinding even ¼ in. thick on the screens.

On page 298 occurs the phrase, "irregularities in pneumatic-table performance caused by intermittent feeding." The authors give a condition 9.8 per cent. float in the refuse at 1.3. Of course, I am not familiar with the operation and with these experiments, but such a figure is outside any experience I have ever had, either experimentally or commercially, in England, once the experiment had been boiled down to a reasonable point. I should like to know the exact conditions under which, with a uniform, constant feed, 9.8 per cent. of float was found in the refuse.

T. FRASER.—I believe Major Appleyard may have misunderstood what we had to say about the effect of moisture on the operation of the tables. We qualified that by saying that it applied particularly to the practice of treating an unsized feed on a table, meaning a size that ranges from a certain maximum down to and including the dust.

The difficulty, of course, is caused by the fine, wet coal or clay material blinding over the holes in the table deck, and I have known some instances of the coal piling up on the table and refusing to move, apparently because of the amount of moisture in the coal. After the condition was corrected, a satisfactory performance was obtained.

K. C. APPLEYARD.—May we have your experience with different types of refuse?

T. FRASER.—In treating American coals, the great difficulty, particularly with dry cleaning, is the amount of middling material, bone and that sort of thing, which, when we attempt to separate it, necessitates rather inefficient operation in order to get as clean a product as we may demand. There may be considerable difference in that respect between American and English coals. I think you will usually find that the great variation in the amount of bank loss or loss of coal that we have in the refuse

is largely due to the nature of the coal, and not only that, but to the specification on the cleaned coal—how clean we want the coal. The middlings that we want to separate very often are the cause of large coal loss in refuse; because of the difficulty in separating at the particular point where the market demands a separation.

J. R. Campbell, Scottdale, Pa.—Mr. Fraser, is not 9 per cent. float in refuse pretty good washer performance in America, on an average?

T. Fraser.—That involves something that is likely to lead to an argument. Many people are a little touchy on the subject of bank loss. I have always considered that under 10 per cent., on an average, of all sizes was pretty good refuse. Usually we do not have much refuse in our bituminous coals. When we get the amount of coal in the refuse under 10 per cent., we have our bank loss down below 1 per cent. which is not a very high loss.

R. MacLachlan.—I believe Major Appleyard is referring to the low percentage of sink in the clean coal.

K. C. Appleyard.—The high percentage of float in the refuse is what I am referring to—9.8.

R. MacLachlan.—That is largely because we have a very small total percentage of float in the refuse. I know of one district which, with other types of refuse and other types of coal, runs over 60 per cent. of float in the refuse. All through this country you will find large variation in the percentage of float in the refuse. It is a question of the quality of the refuse.

H. F. Yancey, Seattle, Wash. (written discussion).—The authors have presented definite data on a subject about which there has been much speculation and guessing. I refer to their table showing the effect of surface moisture on the efficiency of screening fine sizes of coal. This question has often been raised; here we have an answer. In summarizing the paper, six factors necessary to the efficient performance of pneumatic cleaning plants are cited. Five of these factors are applicable, and necessary as well, to the efficient operation of cleaning plants using water instead of air. One of the most important points made in this paper is the conception of regarding the product of the coal mine (where cleaning is required) as raw material which must be processed in a preparation plant to produce a finished article for the market.

Coal Preparation Problems in the Illinois Field

By David R. Mitchell,* Urbana, Ill.

(Pittsburgh Meeting, September, 1930)

This paper discusses some of the fundamental physical and chemical characteristics of coal in Illinois that affect its preparation for the market. At the present time preparation consists almost entirely of sizing and hand-picking. The washing of coal has almost entirely disappeared from the state. This is a rather startling fact when one considers that in the year 1908 Illinois outranked all other states in the quantity of coal washed and a few years later, in 1912, 85 washing plants were reported. Two reasons are apparent for this change. One, which is probably the most important, is that immense areas of comparatively clean coal were opened; the other is that many of the early washing plants were inefficient, so that washed coal was looked upon with disfavor in many trade centers.

Objectives of Operators

Some of the outstanding preparation aims being given due consideration by operating men are:

1. The greatest possible production of lump and domestic sizes which bring in the market at least $1 per ton more than the fine sizes or screenings.
2. Efficient screen sizing with as little breakage as is consistent with economical operation.
3. The removal of extraneous ash and sulfur.
4. Utilization of low-priced screenings.
5. Suppression of dust in domestic coal shipments.

Preparation Underground

To get the greatest possible production of the larger sizes, preparation must start at the face. Improvements in blasting and loading have increased the sizes larger than 2 in. by as much as 20 per cent. at some mines. Tiffany and McKitterick[1] report the results of an investigation they made at a mine in southern Illinois in which, by a carefully controlled

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[1] J. E. Tiffany and J. J. McKitterick: Method of Increasing Lump-coal Production, with Especial Reference to Southern Illinois. U. S. Bur. Mines *Rept. of Investigation* 2697 (1925).

blasting program, the amount of 6-in. lump coal was increased approximately 2 per cent.

Properties of Illinois Coal

The chief physical and chemical properties of Illinois coal may be enumerated as follows:

1. Hardness, which in many of the beds is practically equal to that of anthracite.
2. High ash content—ranging from a low of about 7 per cent. to a high of 25 per cent.
3. High sulfur, ranging from 2 to 6 per cent., except in a more or less restricted area.
4. High moisture. Coals from the southeastern part of the state may have a moisture content as low as 5 per cent., whereas those in the northern field contain as much as 18 per cent.
5. Much mineral charcoal of the soft friable kind present along partings and disseminated through the coal bed.

The very hardness of these coals makes it easy to size them without undue loss due to degradation into finer sizes during screening or conveying.

Screen Sizes Made in Illinois

As more than 50 per cent. of the production goes to the domestic trade, sizing screens have been adopted generally. A recent survey shows that nearly 100 per cent. of all of the mines are equipped with efficient screens. More and more the buying public requires closely sized coal. The sizes generally made are shown in Table 1. These sizes and any combination or modification of them can be made by most tipples. Rescreening has been instituted by a number of mines to ensure a more uniformly sized product.

Table 1.—*Screen Sizes Usual in Illinois*

Trade Name	Size, In.
Lump	Over 6
Furnace	6 × 3
Small egg	3 × 2
Stove or No. 2 nut	2 × 1½
Chestnut or No. 3 nut	1½ × ¾
Pea or No. 4 nut	¾ × ⅜
Carbon or No. 5 nut	⅜ × 0

Improvements in sizing practice have tended toward the elimination of breakage, particularly in the larger sizes. The modern screens are set nearly horizontal and screen out the coarser sizes first. Feeders, chutes and loading booms are constructed so that the coal is conveyed

gently. Although there has been considerable improvement in screening and conveying practice, there is room for still more.

Quality of Lump Coal

Most tipples are equipped so that the large sizes can be hand-picked. It appears, however, that this does little more than improve the appearance of the coal, lump coal being sold on its appearance. Table 2 gives information on 6-in. lump samples, which were collected carefully from loading booms over a period of at least one day and represent closely the average for the mine for the day. At the same time lumps were picked from railroad cars, to get an idea of variation in lumps as to good, medium and poor. Finally, after the sample of coal was received at the laboratory, it was carefully hand-picked to give some information as to ash reduction after careful picking. A study of this table is interesting because it clearly shows the difficulty of judging visually the ash content of a lump shipment. In the case of mine 2, the lumps selected as good had a higher ash content than the lumps selected as average. The

Table 2.—*Variation in Ash and Sulfur of 6-in. Moisture-free Lump Sample*

	Mine 1		Mine 2			Mine 3		
	Ash, Per Cent.	S, Per Cent.	Ash, Per Cent.	S, Per Cent.	Percentage of Total Sample	Ash, Per Cent.	S, Per Cent.	Percentage of Total Sample
Selected good lump	14.2	2.4	5.9	1.2		12.1	3.1	
Selected average lump	15.6	3.1	5.6	0.9		15.8	5.5	
Selected poor lump	24.8	6.9	17.7	0.7		34.6	8.6	
Selected poor lump	18.5	2.9						
Sample of 3000 lb	15.1	3.5	8.0	0.7	100.0	11.7	4.4	100.0
Hand-picked—clean			6.6	0.7	88.2	10.9	3.7	92.0
Hand-picked—refuse			17.8	0.9	11.8	20.2	12.8	8.0

extreme variation in good and poor lumps is also to be noted. This is remarkable because these lump samples were picked from railroad cars and were from coal that had gone over the picking tables. The actual ash reduction from hand-picking is small, considering the amount of coal that had to be discarded, which would lead to the conclusion that picking 6-in. lump coal does little more than help the appearance of the coal.

The discarded lumps were high in ash, although not sufficiently high to cause them to be classed as refuse free of coal. By crushing the discarded lumps and further cleaning, a considerable saving of coal could be effected.

Impurities in Coal

The common impurities are shale, clay, pyrite, calcite and gypsum. Table 3 gives the results of ash analysis of some of these impurities as well as of the coal constituents, bright coal, dull coal and fusain. The samples of impurities were carefully selected in the field. The samples of coal constituents were selected with the aid of a binocular ore-dressing microscope from samples of the coal crushed fine enough to free the constituents from one another. It is apparent at once that fusain for this

Table 3.—*Ash Content of Coal Constituents and Associated Impurities in Western Illinois No. 5 Coal*

Constituent or Impurity	Ash, Per Cent.
Bright coal (anthraxylon)	4.8
Dull coal (attritus)	11.9
Fusain (mineral charcoal)	34.4
Pyrite	59.2
Clay	69.5
Shale	74.2

particular bed should be classed as an impurity. It is also apparent that if bright coal predominates it will be a high-grade coal and if dull coal predominates it will be a low-grade coal.

Effect of Fusain

Impurities are generally concentrated in the finer sizes and must be removed mechanically if these sizes are to be improved. Fusain is usually soft and friable. Clay and some of the associated shales found with Illinois coals are also soft and friable. They are generally concentrated in the finer sizes and their removal constitutes a considerable cleaning problem. In general, coals from this field increase in ash and

Table 4.—*Screen Analyses of Southern Illinois No. 6 Coal*

Size	Percentage of Total	Moisture-free Ash, Per Cent.	Moisture-free Sulfur, Per Cent.
6-in. lump	12.6	8.0	0.7
Inches			
6 × 3	20.3	7.0	0.9
3 × 1¼	17.2	8.3	0.8
1¼ × ¾	15.3	8.6	0.8
¾ × ⅜	5.9	8.5	0.8
⅜ × 3/16	11.5	9.2	0.9
3/16 × 10 mesh	5.8	8.7	0.9
10 × 20 mesh	4.6	8.0	0.8
20 × 48 mesh	4.0	12.4	0.8
48 × 0 mesh	2.8	17.2	0.8
Total and average	100	8.6	0.8

sulfur percentages in the finer sizes. Table 4 is the result of a detailed screen analysis of a southern Illinois coal. This coal is a relatively clean one and the increase in ash in the extremely fine sizes is due almost entirely to a concentration of high-ash fusain in these sizes. This concentration of high-ash fusain complicates the cleaning of these sizes because fusain tends to concentrate in the sludge of a wet-washing plant and in the dust of a dry-cleaning plant. Concentrations of fusain to the extent of 25 or 30 per cent. of the dust or sludge produced in a cleaning plant would not be improbable.

This condition of concentration of fusain in fine sizes is common. However, in most of the coals the increase in ash percentages of the finer sizes begins at a much coarser size than is noted in the example given, on account of the presence of friable bands of shale, clay, pyrite and calcite as well as the fusain. In many coals the increase starts at about ¾ in. and continues down to the finest dust.

Problem of Fine Size

The total production of coal in Illinois in 1929 was approximately 61,000,000 tons. Of this, 17,500,000 tons was screenings or sizes finer than chestnut (1½ by ¾ in.). These screenings, which constitute about 28.7 per cent. of the yearly output of coal, are high in ash and fairly often high in sulfur. They are quoted on the market as low as $1 per ton, which is much below the cost of mining. The larger sizes must carry this differential between the cost of production and the selling price. The problem of how to prepare these screenings so that a satisfactory price can be obtained for them over and above the cost of mining and preparation is a real one, and deserves the best thought of the coal chemist and coal preparation engineer.

It is reported that one of the railroad companies operating a group of mines in the state screens out certain sizes of coal from some of its mines because of the lowered efficiencies of the locomotives when these sizes are fired. It attempts to utilize these "black sheep" screenings in the office buildings and stations during the winter months.

Clean Screenings Could Be Used for Pulverized Coal and Coke

One manufacturing company is buying lump coal and is pulverizing it for powdered-coal firing rather than pulverizing the high-ash screenings available. If cleaned screenings were available, pulverizing costs could be reduced considerably.

A problem faced by coke-oven men is that of getting a low-ash coal for coke. Screenings are undesirable not only because of their high ash content but because of concentration of fusain, which impairs the coking quality. It is reported that one company in the Chicago district which makes a special low-temperature coke for domestic use, buys lump and

egg sizes because of their lower ash and sulfur content. Screenings of low ash content, if too much fusain were not present, could be used as easily and crushing cost greatly reduced.

Effect of Moisture in Coal

Moisture in coal also raises important problems connected with the marketing and cleaning of coal. As previously stated, the water content of Illinois coal varies from a low of 5 per cent. to a high of 18 per cent. This is not moisture-producing wetness, but the normal moisture content of the coal bed, so that such a coal comes from the mine dry and dusty. The moisture varies in transit from the mine to the consumer, generally to the benefit of the consumer because the coal more commonly contains less moisture after shipment than before. Coal purchased on specification, either on basis of heat value or ash content, becomes somewhat difficult to evaluate because of changes in moisture content. Moisture, like ash, is an incombustible and its mining and transport represent a loss similar to that produced by ash; moreover, it cannot be, or at least is not, removed to any extent by preparation processes.

On Testing for Washability

It is, however, to the effect of moisture upon the accuracy of the methods of testing coal for washability that particular attention should be directed. Change in moisture affects the specific gravity of coal particles used in such tests. The writer found as much as 0.15 difference between the specific gravity of air-dried coal and fresh coal. Thus, if float-and-sink determinations were made on dry coal and water-saturated coal, there would be considerable difference in results. A third condition enters into the problem of float-and-sink testing when organic solutions are used. It is common practice to test large sizes with zinc chloride solutions and the finer sizes with solutions of carbon tetrachloride and benzol, or carbon tetrachloride and bromoform. A composite of these results is made to get a picture of the washability of the coal as a whole. The question naturally arises: Is it safe to make a composite, or to compare, the results of float-and-sink determinations on different sizes of coal, using aqueous solutions for some sizes and organic solutions for other sizes?

To obtain some definite information on this phase of the problem, tests were made on a sample of coal having a normal coal-bed moisture of about 15 per cent. The rate of increase in specific gravity of coal particles was noted when they were suspended in water and in organic solutions of carbon tetrachloride and benzol. The final specific gravities of duplicate coal particles after soaking in these liquids were nearly the same, but the organic liquids were absorbed much faster than water. At the end of 20 min. the coal samples in organic liquids apparently had absorbed

all of the solution they could take in. A much longer time was needed for samples suspended in water. Some samples increased in weight for a period of 12 hr. before coming to a stable condition.

Actual float-and-sink determinations were then made on samples of the same coal. The results of one of these tests is given in Fig. 1. Air-dried coal and saturated coal were used with solutions of zinc chloride as the separating medium. Tests were then made on air-dried coal, 3⁄8 by 3⁄16 in., using carbon tetrachloride as the separating medium. The coal was carefully sampled and the results were plotted (Fig. 1). The two samples upon which the zinc chloride solutions were used has an average ash content of 13.0 and 12.8. The sample that was used for the carbon tetrachloride determinations had an average ash content of 11.8 per cent. There was an error of approximately 1 per cent. in cutting out this sample as compared with the other two, which checked almost exactly.

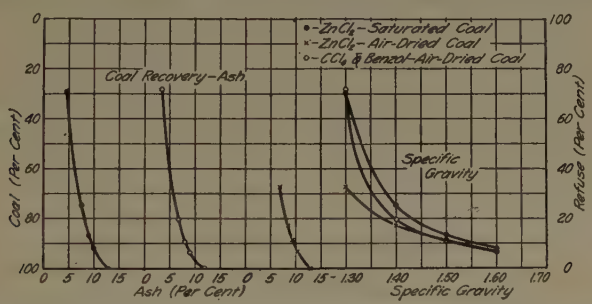

FIG. 1.—COMPARATIVE CURVES OF FLOAT-AND-SINK TESTING OF SATURATED AND AIR-DRIED COAL.

The important consideration is that of recoveries. The curves show that recoveries using carbon tetrachloride solutions on an air-dried coal and zinc chloride on saturated coal agree closely. There is a great difference between the results obtained with zinc chloride on saturated coal and on thoroughly air-dried coal. At 1.30 sp. gr. there is a difference of about 40 per cent.

Although these tests are meager, they point the way to the control necessary if check results are to be obtained in the float-and-sink testing of these coals. Too much emphasis cannot be placed on this phase of testing. The coal to be tested by aqueous solutions should always be brought to the same condition of moisture content before being tested. Moisture determinations should be made and then all results calculated to a moisture-free basis.

Organic solutions are preferable for the fine sizes because they wet them readily and the solution is easily removed from the coal by drying the float-and-sink fractions. The coal should be air-dried thoroughly. The results then obtained by using organic solutions on air-dried coal are

comparable to results obtained by using aqueous solutions on saturated coal. Such results can be combined to get an accurate composite of any group of sizes desired.

Sulfur Reduction

Ash reduction to a certain extent can be accomplished fairly easily in most Illinois coals. Sulfur reduction is another problem and for most of the high-sulfur coals no great reduction is possible. This may not be true at all mines, but is true for a great many. The reason for this is that the organic sulfur content for most of the high-sulfur coals constitutes about half of the total sulfur. Table 5 gives some analyses of the sulfur forms in the No. 2 bed of coal in northern Illinois. About 50 per cent. of the total sulfur is in the removable form of pyritic sulfur. This is an important consideration if a coal is to be used for metallurgical purposes. A detailed chemical analysis of the forms of sulfur in the coal bed may give all the information necessary as to sulfur removal. In any case, it is recommended that chemical analysis of face samples be made for the various forms of sulfur before any extended washability studies are made or before a cleaning plant is erected, which has been done in years past for the express purpose of removing unremovable sulfur.

Table 5.—*Analysis of Sulfur Forms in Illinois No. 2 Coal* (*Moisture-free*)

Mine Channel Sample Number	I	II	III
Sulfate sulfur	0.09	0.07	0.05
Pyritic sulfur	2.75	1.82	1.77
Organic sulfur	1.93	1.78	1.86
Total sulfur	4.77	3.67	3.68

Spraying Coal to Suppress Dust

The problem of dust suppression in domestic shipments is being solved by spraying the coal with solutions of calcium chloride. Several plants are in operation.

Acknowledgments

Special mention should be made of the assistance of Mr. Walter W. Anderson and Mr. J. A. Bottomley, recently graduated in coal-mine engineering at the University of Illinois, for assistance in the laboratory work on float-and-sink testing and specific-gravity studies of coal, some of the results of which are presented here. Also, Dr. Gilbert H. Cady of the Illinois State Geological Survey courteously furnished data on the coal screenings produced during 1929.

DISCUSSION

(*Chester M. Lingle presiding*)

H. F. Yancey, Seattle, Wash. (written discussion).—This paper calls attention to facts which are important in other fields as well as in Illinois. A wide variation exists in the impurity content of fusain in coals generally. Mr. Mitchell's sample from the Illinois No. 5 bed contained 34.4 per cent. ash. Two samples taken from a 2½-in. band (of fusain) near the roof of the No. 6 bed in the Middlefork mine, Franklin County, Illinois, by Thomas Fraser and me[2] showed in one 39.3 per cent. ash and 6.6 per cent sulfur, and in the other, 32.3 per cent. ash and 20.9 per cent. sulfur. An interesting fact was that virtually all of the sulfur was present in pyritic form in both samples. On the other hand, Holbrook[3] and Savage[4] have given analyses of fusain from the No. 6 bed in Williamson County, Illinois, which showed less ash than was normally present in the adjacent coal, or about the same amount. Examination of samples of fusain from the Pittsburgh bed, made at the Pittsburgh Station of the Bureau of Mines,[5] has disclosed two varieties, one called "soft" and the other "hard." The hard variety contained a high percentage of calcium carbonate. J. B. Morrow has full information on this occurrence. Fusain is porous and hence may contain as an impurity various materials, depending upon the composition of the underground waters which come in contact with it.

The author's experiments with the combination of float-and-sink data for the coarse and the fine sizes of coal into a composite are interesting. A study of this problem has also been made by Bird and Messmore[6] at the Northwest Experiment Station of the Bureau of Mines. They recommend as a workable and approximate correction that the organic solutions used for air-dried coal be slightly higher in density than the aqueous zinc chloride solutions used for coarse coal. In most cases, however, the discrepancy involved in combining float-and-sink data on coarse and fine sizes of coal where different methods of separation and moisture conditions are employed for the respective sizes is not of practical importance, because the fine sizes separated on organic solutions generally constitute a minor proportion of the total composite sample. Moreover, the error that may be introduced into the composite yield-ash curve is generally much less than any usable interpretation which may be placed upon the results.

J. R. Campbell, Scottdale, Pa.—My practical experience with the Illinois coal is limited. When I was with the United States Steel Corpn. we had a washer in Franklin County, washing the coal for metallurgical purposes. The product was shipped to the Joliet and Gary plants, for mixture with eastern and southern coals for making coke.

However, in the line of my present work, we have made a good many washability studies of Illinois and Indiana coals. As a general rule, they do not present as attrac-

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[2] H. F. Yancey and T. Fraser: Distribution of Forms of Sulphur in the Coal Bed. Univ. of Ill. Eng. Expt. Sta. *Bull.* 125 (1921) 37.

[3] E. A. Holbrook: Dry Preparation of Bituminous Coal at Illinois Mines. Univ. of Ill. Eng. Expt. Sta. *Bull.* 88 (1916) 62.

[4] T. E. Savage: On the Conditions Under Which the Vegetable Matter of the Illinois Coal Beds Accumulated. *Jnl. of Geol.* (1914) **22,** 761.

[5] J. D. Davis and W. D. Pohle: Effect of Fusain and Related Inerts of Pittsburgh Coal with Particular Reference to Coking Properties. Presented at Cincinnati Meeting, Amer. Chem. Soc., Sept., 1930.

[6] B. M. Bird and H. E. Messmore: The Float-and-Sink Testing of Fine-Size Coal. Univ. of Wash. Eng. Expt. Sta. *Bull.* 46 (1928) 13–18.

tive a product as do the eastern coals. Mr. Mitchell has brought this point out very clearly. If 8 or 9 per cent. ash can be obtained in the Illinois coals by washing, it is very well. The improvement in sulfur by washing is slight. There seems to be a great deal of organic sulfur in the Illinois coal, and being organic it cannot be removed by washing.

As compared with most of our eastern coals, it does not have the flexibility of markets. The coal is higher in ash, sulfur and moisture. It is, however, the coal that has been laid down in that section and is mined and sold extensively in certain markets.

Based upon washability studies there is no question that the coal can be beneficiated by washing. In many cases the ash in the average mine product can be reduced 5 to 6 per cent. The Illinois coal as mined seems to have a wide range in ash, especially in the finer sizes. Some cars of unwashed coal may show from 6 to 8 per cent. ash increase over the average. Washing here will do a great service in insuring a uniform product.

We are striving for and attaining low ash and low sulfur in the Pittsburgh district and we must recognize that comparative ash and sulfur cannot be obtained except in a few cases from Illinois coals. That, however, does not preclude the fact that the cleaning of Illinois coal should be a profitable venture.

Operation of Rheolaveur Plant at Dorrance Colliery, Lehigh Valley Coal Co.

By Edgar Schweitzer,* Wilkes-Barre, Pa.

(Pittsburgh Meeting, September, 1930)

The original Dorrance breaker of the Lehigh Valley Coal Co. was erected in 1883. The coal beds were clean and dry, consequently a dry preparation system was used, consisting of revolving cylindrical screens for sizing and hand-picking, to remove the few impurities from the run of mine coal. The picking or cleaning room contained long parallel rows of chutes with seats mounted on them, for the use of the breaker boy, who bent over the chute and picked out the refuse material from the coal as it gravitated down the chute. The introduction of dry mechanical pickers, of the sliding frictional type, in the early nineties lessened the number of boys, but hand-cleaning on the coal and refuse discharge ends of the mechanical picker was necessary. The Pardee spiral pickers, which were brought out about 1900, were better than the gravity pickers and gradually replaced them. In 1912 the breaker was entirely remodeled, although the original frame was retained. The revolving screens were replaced by shaking screens, and Pardee spirals were installed on all sizes of coal from egg to pea coal, inclusive. Hand-picking still was necessary on the coal and refuse ends of the spirals, although the changes did result in a reduction of the former breaker force.

As mining advanced, and the thick clean bed became exhausted, it was necessary to mine veins containing a high percentage of impurities, which complicated the preparation problem. The most serious difficulty was the increase in the quantity of damp or wet run of mine, which could not be fully cleaned on the spirals. When this quantity exceeded approximately 5 per cent. of the feed, it was necessary to send the excess of the wet coal to the company's Prospect colliery for wet washing. This plan worked well for a time, but as the amount approached 15 per cent., with the prospect of a further increase, the company considered the building of a wet-washing plant. The Dorrance breaker was the only dry-cleaning plant operated by the company. At the seventeen other collieries, wet washing was employed, and jigs, both plunger and pan, were used. All the practical methods used in the anthracite fields were investigated, and it was finally decided to adopt the Rheolaveur system. The decision was due largely to the experience gained from the the operation of a Rheo-

* Assistant Mechanical Engineer, The Lehigh Valley Coal Co.

laveur experimental plant at the Hazleton shaft breaker, run for two years in parallel with the jig plant in the breaker. The results at the experimental plant showed an increase of approximately 2.2 per cent. of the prepared sizes over the jig plant, equivalent to approximately $35,000 per year, due to increased yield alone, and also an estimated saving in labor of about $30,000 per year. It was estimated that the first cost of the Rheo plant would be less than that of a jig breaker and as cheap as any other system, and, in addition, the operation cost, as well as the loss due to degradation, would be no greater and probably less than the other systems investigated.

An analysis of the probable run of mine feed was made by selecting mine cars from each of the seven veins mined, and, by visual inspection, the products larger than egg or +3⅜-in. were divided into coal, bone and refuse. Sizes larger than egg were broken down to egg and smaller, and a composite sample prepared, taking quantities proportional to the tonnage being mined from the respective seams and proportional to the actual weights of corresponding material in the mine cars. The sample was screened into sizes and float and sink tests conducted up to 2.00 sp. gr. Indications were that a recovery at 1.90 sp. gr. would be practical and result in approximately 10 to 12 per cent. ash coal. After the laboratory tests, the samples were submitted to a company coal inspector to be hand-picked, in order to determine exactly what material he would accept as coal and reject as refuse. It was interesting to note that the inspector's results checked within 1 per cent. of the sink and float tests, and that a recovery at 1.9 sp. gr. would, in his opinion, produce a satisfactory combustible product. In practice, it has been found necessary to reduce that limit, due to an increasing critical market, to around 1.65 to 1.70 sp. gr. This desire on the company's part to manufacture a "quality anthracite" instead of an "anthracite combustible" results in an estimated loss of 39,000 tons of combustible material, in the domestic sizes only, based on 1929 tonnage, worth approximately $300,000.

The Rheo plant is designed for an output of 3000 tons per 8-hr. day. The run of mine feed contains approximately 8 per cent. of slate refuse plus the light-gravity material from 1.70 to 1.90, consisting of bone coal, laminated pieces, half and half and capped. However, mine cars from certain sections contain as much as 25 per cent. refuse, which is largely laminated bone. Some light bone, condemned for appearance, which floats at 1.58 sp. gr., is lighter than the coal and is difficult to wash, therefore when there is more than the amount allowed in broken and egg sizes it must be hand-picked from the coal.

Plant Flow

Fig. 1 shows the flow through the plant. The coal is conveyed to the breaker by means of a 42-in. run of mine conveyor with 425 tons per hr.

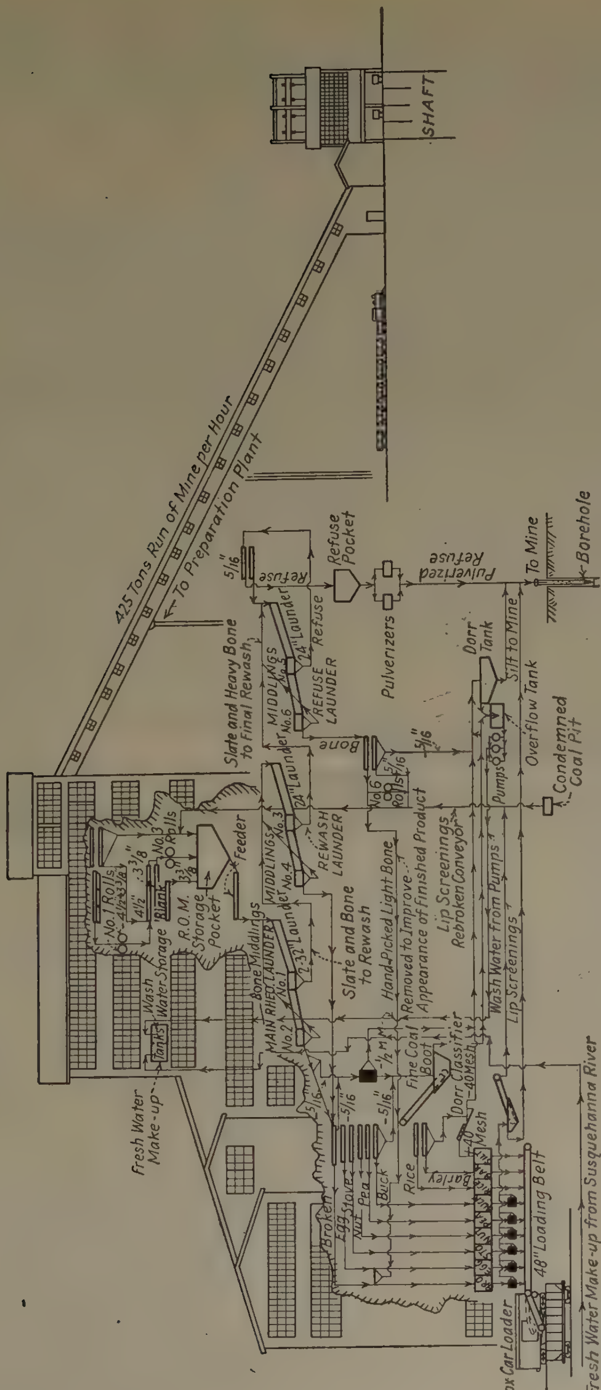

FIG. 1.—FLOW THROUGH RHEOLAVEUR PLANT, DORRANCE COLLIERY.

capacity. The usual separation of the breaker is made at the top through a lump and steamboat shaker and the coal passes over a picking table, in which the tramp iron and pure rock only are removed by a couple of boys. The coal passes over the intermediate shakers, the broken and egg coal passing through two sets of No. 3 rolls, and the entire mass passing into a storage pocket. Out of this storage pocket there is a reciprocating feeder which discharges the coal into the main launder. The feed launder is about 18 ft. long, and the distance between the boxes is about 13.5 ft. The product, dropping through No. 1 box, is discharged through the Rheo box into a rewash launder and the product dropping through No. 2 box, which is a mixture of coal, bone and some slate—laminated material—is taken up in an elevator and discharged back into the feed again. The product coming across the main launder, as well as the first rewash launder, is sent direct to the sizing shakers. The $-5/16$ material is discharged through a screen off the main launder and also off the broken shakers, passed over the dewatering screen and down into a fine coal plant.

The material through the dewatering screen goes to the Dorr tank, and the same process is repeated all of the way through the plant, the material dropping through No. 1 box going back into the rewash and the material dropping through No. 3 box going to a final rewash launder. The material in No. 4 box is sent back into feed again which gives an opportunity to rewash it. As mentioned before, the material off the end of the rewash launder is sent directly to the pocket. It is supposed to be pure coal, and generally is.

In the final rewash launder, the material coming through No. 5 box is sent direct to the rock pocket through the pulverizers and down into the mines. The material dropping through No. 6 box is sent back over a set of shakers, and the $-5/16$ material is put back into the feed to help create bed. The extremely heavy material discharged over the end of the final launder is in sizes from $+3/4$ in., or, say, nut coal on up, and is discharged over a "bone shaker" and through the rolls; it is elevated back into the conveyor line into the main feed and goes back into circulation.

Water used for washing is returned to the Door thickener, from which it overflows into the circulating pump sump. The only water lost is the water used to wash the coal during loading into railway cars, about 400 gal. per min., the underflow from the Dorr thickener, and a small amount used in the refuse pulverizers. The make-up, amounting to 1200 gal. per min., is pumped from the Susquehanna into a fresh-water storage tank and is used only to spray water on the sizing shakers and lip screen sprays. Because of the conditions noted below, the control of the circulating and make-up water is an important part of breaker operation. Dorrance colliery is located within the City of Wilkes-Barre. Land is valuable, and, as there is none available for outside refuse storage, the entire discharge from the breakers must run into the mine through a borehole,

the capacity of which is limited to 1500 gal. per min., so that any outflow in excess of that amount causes an overflow at the hole. The excess quantity would run into the Susquehanna River, which must be avoided.

Breaker Structure

The breaker structure is steel with asbestos sides and roof and steel sash. The capacity is 616,000 cu. ft. and 540 tons of steel. Note the comparison with a jig breaker having an estimated daily capacity of 2500 tons, containing 1,424,240 cu. ft. with 1315 tons of steel, and another breaker of the same capacity equipped with the Chance sand flotation process, containing 1,080,000 cu. ft. and 1020 tons of steel.

Construction was started in May, 1929, but, due to the fact that the old breaker could not be abandoned until the new one was started, and also because the new plant was to occupy a portion of the old breaker site, construction was slow. The new breaker took part of the coal on Jan. 16, 1930, but it was not until May 15, 1930, that the plant was completed.

TABLE 1.—*Labor at Dry and Wet Breakers*

	Dry Breaker	Rheo-laveur Plant		Dry Breaker	Rheo-laveur Plant
Preparation			Breaker Maintenance		
Breaker boss	1	1	Carpenters	2	0
Ticket taker	1	2	Refuse		
Dumpers	2	2	Machine attendant	2	2
Plate men	13	2	Laborer	1	1
Pickers, pure coal	7	3		—	—
Pickers, refuse	5			3	3
Machinery operators	2		Transportation		
Machinery attendants	2	7	Engineers	11	11
Oilers	2		Locomotive engineers	4	2
Picker boss	1		Headmen	9	0
Cleaners	2	2	Runners	3	0
Engineers	1		Oilers	3	0
Laborers	4	2	Track repairmen	2	2
	—	—		—	—
	43	21		32	15
Loaders			Total	95	52
Box-car loaders	4	1	Reduction	43	
Coal-car loaders	4	2			
Car runners	1	3			
Car cleaners	4	5			
Retail	1	1			
Boss	1	1			
	—	—			
	15	13			

Labor

The most direct gain by the Rheo breaker over the old dry breaker was the reduction in labor of 43 men and a decided increase in yield. Table 1 gives a comparison of the labor forces.

The seven men on preparation listed as machinery attendants are in charge of the Rheo plant and tributary machines. They are all paid the same rate and are assigned to no particular position, but, since rates are equal, they may be shifted from one job to another without any trouble. This is a new arrangement but is working out very well.

Grades of Coal

It is rumored that Dorrance coal is easy to prepare, but the contrary is the fact. There is much light bone or laminated material that is lighter than coal, which passes through the Rheo launder, and there is no gravity machine that will remove light material into the refuse and pass heavier pieces into the coal. The coal inspector classifies coal as First, Second and Third grades, and he will accept only a small amount of the second and less of the third grade for the railroad cars. In order to produce "quality anthracite" it is necessary to hand-pick broken and egg

Table 2.—*Grades of Coal and Refuse in Feed*

Grades	Float, Per Cent.					Sink, Per Cent.	Total, Per Cent.
	1.60 Sp. Gr.	1.60 1.65 Sp. Gr.	1.65 1.70 Sp. Gr.	1.70 1.75 Sp. Gr.	1.75 1.90 Sp. Gr.	1.90 Sp. Gr.	
Egg							
No. 1 coal	60.29				0.14		60.43
No. 2 coal	13.18	1.71	0.7				15.59
No. 3 coal	4.00	1.44	2.22	0.21	0.16		8.03
Half and half	0.58	0.72	0.53	2.08	3.44		7.36
Bone	0.19		1.85		0.53		2.57
Slate					1.14	4.89	6.03
Stove							
No. 1 coal		66.28	0.39			0.05	66.72
No. 2 coal		8.68	1.72	0.08			10.48
No. 3 coal		1.48	1.37	0.44	0.24		3.53
Half and half		3.11	0.73	0.65	0.46		4.95
Bone		0.28	1.92	0.89	0.57		3.66
Slate				0.75	0.37	9.54	10.66
Nut							
Coal		89.13	0.71				89.84
Half and half		0.73			0.77	0.29	1.79
Bone		0.14			0.72	0.05	0.91
Slate					0.27	7.19	7.46

sizes to remove pieces that have a poor appearance although they would not impair the burning qualities of the coal. Table 2 gives percentages of the different grades. This table shows the overlapping of the various grades and indicates the washing difficulties. The second and third grades of coal are of good quality but lack the luster or brilliancy of the first grade coal. There is only a slight difference between the grades and it is extremely difficult to classify them correctly; only an experienced inspector can do that.

With the various grades to contend with and the perfection demanded by the trade, in appearance particularly, while the plant was being adjusted to produce satisfactory coal as well as a low percentage of coal in the refuse, condemned coal was to be expected. The percentage has been reduced month by month, until now it is almost negligible. There are days when the run of mine feed is high in bone coal, and on these days the condemned coal is up, but on the whole the amount is satisfactory. Table 3 shows the percentage of condemned of the total loaded by months and Table 4 the number of cars condemned and the cause. The standards used by the Inspection Department are given in Table 5. Table 6 shows the percentage of coal in the total refuse by

TABLE 3.—*Percentage Condemned of Total Loaded by Months*

Month	Railroad Cars Loaded	Condemned, Per Cent.		
		For Slate	For Slate and Bone	For Bone
January	279	0.72	1.61	6.63
February	995	0.70	2.96	5.07
March	871	0.46	3.85	11.71
April	599		1.09	2.92
May	1130	0.09	0.27	3.27
June	1150		0.30	0.87

TABLE 4.—*Cars Condemned for Various Causes*

Month	Railroad Cars Loaded	Number Cars Condemned					
		For Slate	For Bone	For Oversize	For Undersize	For Breakage	For Appearance
January	279	6½	18½	4	2	1	0
February	995	36½	50½	10	13	3	0
March	871	37½	102	0	3½	1½	
April	599	6½	21		4	0	3
May	1130	3	35	0	6	7	12½
June	1150	3½	8½	5	11	5	1

sizes. The loss of coal in the refuse is of utmost importance, and is sampled daily.

TABLE 5.—*Percentages of Slate and Bone Allowed by Inspection Department*

Size	Effective 2/19/30		Effective 3/17/30	
	Slate, Per Cent.	Bone, Per Cent.	Slate, Per Cent.	Bone, Per Cent.
Broken	1.0	1.0	1.0	1.0
Egg	1.5	2.5	1.5	2.0
Stove	2.5	4.0	2.0	3.0
Nut	3.0	4.0	2.0	4.0

TABLE 6.—*Percentage of Coal in Total Refuse*[a]

Month	Broken		Egg		Stove		Nut	Pea	Buck	Rice	Total	
	Coal	Chipped Coal	Coal	Chipped Coal	Coal	Chipped Coal					Coal	Chipped Coal
January	0.195	0.302	1.28	0.79	1.35	0.49	1.69	0.79	0.18	0.074	5.56	1.58
February	0.158	0.17	1.30	0.79	1.01	0.49	1.18	0.77	0.28	0.15	4.86	1.46
March	0.078	0.125	1.80	0.92	1.84	0.63	3.89	2.32	0.93	0.52	11.405	1.68
April	0.029	0.002	1.21	0.73	0.95	0.29	2.30	1.86	1.07	0.64	8.07	1.02
May	0.00	0.008	1.08	1.19	0.89	0.63	1.43	1.09	0.66	0.66	5.82	1.83
June	0.00	0.001	0.21	1.02	0.53	0.57	1.52	1.38	0.88	0.68	5.21	1.59
July	0.00	1.08	1.31	2.94	1.92	2.31	4.67	17.1	21.4	24.00		

[a] As the total refuse represents 6 to 8 per cent. of breaker feed, 5 per cent. coal in refuse represents a loss of 0.3 to 0.4 per cent. of coal in feed. Further inspection department tests show that refuse contains practically no first grade coal in prepared sizes.

TABLE 7.—*Power Consumption and Tonnage, Dorrance Breaker*

	March, Kw-hr.	April, Kw-hr.	May, Kw-hr.	June, Kw-hr.	July, Kw-hr.
Group 1, head house	27,300	21,300	31,700	26,800	28,200
Group 2, washer, incl. river pumps	45,800	40,800	61,700	56,200	55,000
Group 3, screening and loading	12,100	9,700	16,100	15,100	15,500
Total kilowatt-hour used, 440 volts	85,200	71,800	109,500	98,100	98,700
Tons shipped	35,231	34,629	56,722	53,259	55,477
Kilowatt-hour per ton	2.42	2.08	1.93	1.84	1.78
Total kilowatt-hour, 11,000 volts	90,150	81,400	114,200	100,500	99,800
Cost per kilowatt-hour	1.05	1.073	0.922	0.978	0.967
Demand kilowatts	600	600	600	600	600
Load factor (24 hr.)	21.1	18.9	26.5	23.3	23.0
Breaker starts	19	17	26	23	24

Power

The plant is electrically operated, using purchased power and individual drives. The total connected load is 1100 hp., including the pump that furnishes make-up water (Table 7).

DISCUSSION

(Paul Sterling presiding)

C. Evans, Jr., Scranton, Pa.—I would like to ask Mr. Schweitzer a question about the storage pocket ahead of the launder, which is an unusual feature, to me at least, in anthracite design. How large is that pocket?

E. Schweitzer.—It is a 200-ton pocket.

C. Evans, Jr.—A 200-ton pocket for all of the launders or for each?

E. Schweitzer.—For the main launder only.

C. Evans, Jr.—How many launders go out of that 200-ton pocket?

E. Schweitzer.—It is a continuous feed.

C. Evans, Jr.—I thought you had two 32-in. launders from the 200-ton pocket. Was that pocket a part of the original design?

E. Schweitzer.—Yes.

C. Evans, Jr.—A pocket is something we have never put in, and I am interested.

E. Schweitzer.—It allows a great deal of flexibility and you are not so dependent upon delays in the shaft. Of course, we are directly connected to our shaft, and it takes care of any variation in our hoist. It also gives us a better mixture in the run of mine. Occasionally we get a bad run of particularly bony coal, laminated coal, which is extremely hard to clean in any kind of a plant, and it reduces the percentage considerably if you can dump it into a large pocket.

C. Evans, Jr.—Do you actually find that a 200-ton pocket is sufficient to afford any useful degree of mixture?

E. Schweitzer.—It seems to with us. I really have nothing to base that on as far as any tests are concerned, because really there is no way of determining it unless you eliminate the pocket entirely and see what the results are without it.

C. Evans, Jr.—The reason I ask is that as your material comes in continuous flow from the mine, it must continually drop into the pocket. If you get 20 mine cars of bad material, those 20 mine cars must be deposited in one mass into the pocket, and must come from the pocket largely in the same mass.

E. Schweitzer.—In a case like that, you will not account for any mixture in the pocket. We do not get all of our bad materials from both shafts. We are hoisting in both shafts, so that it is not so serious a problem with us.

C. Evans, Jr.—You would get that degree of mixture without the pocket.

E. Schweitzer.—Yes, you really can get it on the belt, but in addition to that you have the returns from the No. 6 box coming back, which is rather poor material.

B. H. Stockett, Minersville, Pa.—In Table 6 you show refuse for June of 5.21 coal. In July is there no refuse at all, or no coal in the rock?

E. Schweitzer.—Yes. I explained that in the paper. In July the results in each size were for 100 per cent., whereas in a month like June all sizes were considered to be 100 per cent., so that the total of all sizes is 5.21.

B. H. Stockett.—Would June and July be about comparable in operation?

E. Schweitzer.—As a matter of fact, July was somewhat better.

B. H. Stockett.—Assuming that June is a fair month, with a total of 5 per cent. of coal in the refuse, are not your modern jig breakers doing as well?

E. Schweitzer.—The only way I can answer that question is to say that, of course, we must accept these things as they are. If you actually saw that material and tried to sell it to a customer, you would decide it was not quite as bad as it looked. We might as well be fair, and as long as that is the figure that we are charged with, you must accept it. But we certainly have the material. If we got that on to a railroad car, we would have more trouble with it.

B. H. Stockett.—Who makes the inspection on the coal and the refuse? The same inspection department?

E. Schweitzer.—The same inspection department.

B. H. Stockett.—Would they not make it on the same basis as on the refuse in the coal?

E. Schweitzer.—Well, yes, they do, but on the other hand, getting back to this peculiar separation that our inspectors are making of first, second and third grade coal, I would say that you would not find any that equals the first grade coals in those percentages. It is generally the material that is hardly acceptable in any quantities at all. I know the figures here look bad, and it is hard to explain. The best way to explain it is to see it.

J. F. McLaughlin, Scranton, Pa.—In June on that same refuse table, we note, say in buckwheat, there was a loss of 0.88, and in pea coal 1.38 and in nut 1.52. Buckwheat in July shows 21 per cent. loss. How would you account for that remarkable increase from 0.88 to 21?

E. Schweitzer.—I repeat what I have already said. In a composite sample 21 per cent. of the total buckwheat refuse was coal. In the other case, each size was considered 100 per cent. in itself.

C. Evans, Jr.—Might I ask you a question along that same line? Do you consider that a loss of 21 per cent. in the buckwheat going to the bank is reasonable? You show that in July, of the buckwheat size that went to the bank 21.4 per cent. was coal, as I read your figures, and likewise of the rice size 24 per cent. was coal, and of the pea size, 17 per cent. There is a radical jump in the percentage of coal, in the sizes, as you go down the scale. You indicate that only 4.67 per cent. of the nut size went to the bank as coal, and yet the next smaller size jumps to 17 per cent., which, as I read the figures and as I visualize your operation, shows that you are grinding down your sizes larger than nut very radically.

E. Schweitzer.—We are not doing that in any particular breaker.

C. Evans, Jr.—You are not? Why?

E. Schweitzer.—I do not know that I can explain that. It is high, I will admit. The percentage of pea, buckwheat and rice compared with the other coal, of course is remarkably small. Neither would that hold good in all of our breakers. That is a rather high percentage, but, as I said, the bulk of it is objectionable.

P. Sterling, Wilkes-Barre, Pa.—I might answer that question, Mr. Evans, by saying that they are grinding down, and the material which is passed around is in the neighborhood of 30 tons an hour. The material going around is what the inspector calls second and third grade coal, which is coal that will not pass inspection on the car and that cannot go into the refuse. There is a small percentage of good coal.

C. Evans, Jr.—Do you mean that you are grinding down 30 tons of coal an hour, or 30 tons of material?

P. Sterling.—Of material.

C. Evans, Jr.—That is a relatively small amount in a breaker of that size. In our Chance cone breaker we grind a very much larger amount per hour, in a 2200 or 2500-ton breaker.

W. H. Lesser, Frackville, Pa.—Table 6 indicates that the coal in the refuse is 5.21 per cent. Is that good enough to pay you to break it down and then put it into the system again?

E. Schweitzer.—I doubt it very much. As I said before regarding that material, while we are charged with it and it looks like a total loss, really we are very willing to let it go as a total loss. I do not think it is good enough to smash up. The same thing is true of this 30 tons-an-hour material that passes around, which is ground down. That is heavy stuff and part of it can be reclaimed but it is a rather small percentage.

J. F. McLaughlin.—On page 312, regarding the experimental plant at Hazleton, was it determined that there was an increase in prepared yield of 2.2 per cent.? another saving of $30,000 a year? Was that on the basis of 1.9 separation? How does that compare with your present-day performance since your radical change in July?

E. Schweitzer.—Of course, I mention that as one of the inducements for making this change. The Hazleton shaft was a wet breaker, but in our particular case Dorrance was a dry breaker. In changing from a dry breaker to a wet breaker, there was such a tremendous difference that it is hard to give you any comparative figures. There has been a decided improvement, and the yield is considerably higher, as you would naturally expect. The losses incidental to the old dry preparation were a great deal more than even we realized. We had more going back to the mine than we had supposed. So our increase in yield at Dorrance, due to this change, has amounted to about 6 to 8 points. We get a prepared yield there of around 72 per cent.; that is, based on prepared sizes. Based on the mine car, we get about 66 per cent., and we never had anything like that before. It is pretty hard to compare them, but we were justified in making the change because our losses were so great.

J. Griffen.—Is it true that your prepared yield on the old breaker compared very favorably with your jig breakers?

E. Schweitzer.—Yes, and even on our jig breaker we think we are getting pretty good results and we are fairly well satisfied. Within a few points, taking those jig breakers right in the same neighborhood, the prepared yield will run about 1.87, and the old dry breaker ran around 1.84 or 1.85.

J. S. Miller, Lansford, Pa.—Do you attribute your increase in yield to the Rheolaveur or to a change in mining methods?

E. Schweitzer.—That is a fair question, but the management and the engineers ought to get some credit. It is undoubtedly true that the change in the system brought it about. We have not made any radical change in mining and it is due primarily to the difficulties that we encountered in getting the wet coal that we changed to the wet system. When once in a while, to be perfectly frank, there is a run of exceptionally bad coal, which gives a great deal of condemned coal, our coal inspectors may stretch the limits slightly for a day or two, to avoid congestion in the breakers, but if the trouble continues the condemned coal is put back in the feed again. It depends altogether on the quantities of that material. Last week, for instance, we had only one car of coal condemned and that was for chippings.

G. V. Woody, Pittsburgh, Pa.—I would like to answer Mr. McLaughlin's question as to whether your company's decision to put Rheolaveur in at Dorrance was based upon operating the Hazleton shaft Rheolaveur plant at a 1.90 sp. gr. of separation. I believe the decision was based upon the character of the washed and shipped coal and the small amount of coal in the refuse rather than upon the gravity at which separation was made. As a matter of fact, this Hazleton shaft plant was operating at a gravity somewhat below 1.90, perhaps in the neighborhood of 1.75 and lower at times. Condemned cars were the exception there and I believe the Lehigh Valley inspection is as rigid as any in the Anthracite Region. This statement is based upon experience with coal inspectors in testing coal at a number of operations.

I understand that the chief inspector of the Lehigh Valley Coal Co. indicated that a large portion of the material that is called coal in the refuse at Dorrance would not be permissible in any quantity on a car of coal to be shipped. What are you going to do with material if it cannot be put on a car to be shipped and cannot be put out with the refuse without causing the refuse to be condemned as poor?

C. Evans, Jr.—Burn it in the boiler plant.

G. V. Woody.—That is good, but have you one? I would like to say another thing about Dorrance coal which can be verified either by personal inspection or by looking at inspectors' reports; that is that there are a great many cars of egg coal shipped, I would say more than 50 per cent., on which the inspector's reports show zero slate. During the inspection of the Dorrance coal, "cappy" or laminated pieces are chipped. In other words, if there is a large piece of coal with a small piece of slate sticking to it, the slate is chipped off and called slate in the car of coal to be shipped. Even with this inspection, the statement which I made relative to a number of cars, showing no slate is a fact. Did you bring out in your discussion what your inspection limits now are?

E. Schweitzer.—Yes, I read them. I think they are as low as any, and on this breaker they are slightly lower. The broken was 1 and 1 and egg was 1.5; the stove 2 and 3, the nut 2 and 4. Those are lower limits than those of either the Glen Alden or the Hudson Coal Company.

G. V. Woody.—I understand that both those companies permit the cars to be condemned on account of appearance even though they pass the inspection limits on slate and bone which they have set up.

E. Schweitzer.—We do that, too. In fact, all companies condemn cars on appearance although they may have zero slate and zero bone.

G. V. Woody.—I would like to say another word about the seemingly high percentage of coal in the refuse at the Dorrance plant. In a Rheolaveur plant designed to

handle a coal containing a small amount of fine sizes in the feed, and where these small sizes are consistently clean as they are fed into the plant, it is necessary to circulate an increased amount of fine slate in the system in order to prevent some of the fine coal from going out with the larger refuse. In this plant, the percentage of the fine sizes in the refuse is so small, averaging only 1 or 2 per cent. of the refuse, which is about 6 per cent. of the feed, that it was not deemed necessary to go to any great expense to recirculate the fine heavy slate in order to prevent this small loss in the fine sizes.

P. Sterling.—Mr. McLaughlin, I do not think your question has been answered exactly, as compared to the Hazleton shaft plant under 1.90 sp. gr. recovery. We do not recover at 1.90 sp. gr. It was 1.75 sp. gr. The coal is heavier gravity coal and runs as high as 1.72, but the Hazleton shaft of two years ago is not comparable with Dorrance today because our inspection limits have been reduced. In other words, we would not show 2.2 per cent. recovery today as compared to two years ago.

We have discussed the preparation of coal by two methods, the Chance and the Rheolaveur system. What the operators are trying to do is to turn out a superior product to the trade. They are not sinking and floating the refuse. They are putting out a coal that looks good to the trade and that the trade will buy. That is what they want and that is what the operators have to do. The time may come when some of this coal that is being thrown away today will be sold to the trade as good combustible coal.

Control of Chance Cone Operation

By John F. McLaughlin,* Scranton, Pa.

(Pittsburgh Meeting, September, 1930)

The installation of the Chance flotation system for the preparation of anthracite demonstrated the need for some means by which the specific gravity of the fluid mass in the separating cone could be determined with a fair degree of accuracy. The means used (with the original cones) was a set of wooden balls attached to ropes or chains, each with a different specific gravity. In the Northern Anthracite field, where the cones are generally operated with the specific gravity at 1.65, these gravity balls are provided with a range of 1.6, 1.65, 1.7 and 1.75.

The operator of the cone from time to time would place one or more of these balls in the cone and estimate its specific gravity by the action of the balls. The margin of error with this system was very great, because there was the possibility that the test ball might be held in suspension and carried around by material floating beneath the surface of the cone in what is known as "island" formation—a formation resulting from a low sand line in the separating cone.

The inaccuracy of this system of determination of specific gravity was indicated also by periodic tests of the final refuse bank, which showed excessive losses of coal to the refuse. Further evidence of the unsatisfactory operation of the cone was the uneven appearance of the market product, which caused spasmodic condemnation.

Maintenance of Even Preparation

With the critical market condition of today, it is essential that the preparation units operate consistently to maintain an even preparation. If a plant wishes to maintain a 1.65 specific gravity preparation, which means that all material of a specific gravity greater than 1.65 is classed as "reject," if the specific gravity of the fluid mass runs to 1.70, the market product will not be satisfactory; or if the specific gravity of the fluid mass is 1.60, the market product will be entirely satisfactory but the losses at the slate ends will be excessive.

The desired specific gravity of the fluid mass in the Chance cone is maintained by agitating the sand within the separating cone by means

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* Assistant Superintendent of Preparation, Hudson Coal Co.

of the mechanical agitators, which rotate at a fixed speed, and by means of a flow of agitation water introduced into the cone at any one of four points, which, in the order of their importance, are as follows:

1. Classifier column water line.
2. Bottom agitation spray line.
3. Center agitation spray line.
4. Top agitation spray line.

It is obvious that any increase in the volume of agitation water tends to decrease the specific gravity of the fluid mass, and vice versa. The flow of agitating water is controlled by separate valves to each of the outlets, and the operator soon learns to know how much each of the valves should be opened in order to furnish the volume of water necessary. He is not positive at all times, however, as to whether the sprays which discharge water from these valves are free from obstructions, therefore he cannot know from the position of the valves whether the required amount of water is entering a cone at the proper places. The result is uneven preparation, with perhaps condemnation, particularly if the preparation plant consists of a number of separate units all discharging into the same car. In such a case, a variation in the specific gravities of the several units results in a spotted condition in the market product, which frequently causes condemnation, when perhaps a full carload of the poorest material in the car would pass without question.

Percentage of Sink and Float in Slate Determining Factor

All of these conditions indicated the need of a more positive means of control, and to obtain this control, it was assumed that in order to obtain samples of the final slate and place them in a solution having a specific gravity similar to the desired operating gravity, the percentage of sink and float in the slate end would determine the gravity of the fluid mass in the separating cone; *i. e.*, supposing that 1.65 gravity was being maintained in the separating cone, the solution to test the final slate should have a gravity of 1.65. Numerous tests conducted along this line indicated that the margin of error in this method was between 0.5 and 1.5 per cent.; that is, when the percentage of float in the final slate averaged between 0.5 and 1.5 per cent., the gravity was 1.65 in the separating cone. When the percentage of float increased above 1.5 per cent. the specific gravity in the separating cone was below 1.65. If the percentage of float decreased below 0.5 per cent., the specific gravity in the separating cone was above 1.65. This formed a basis of operating the Chance cones. A Delatester, with a zinc chloride solution, is being used to determine the percentage of float in the final slate that is sent to the refuse bank.

The final slate averages approximately 80 per cent. chestnut, 6 per cent. pea, and 15 per cent. buckwheat and smaller sizes. Egg and

stove size refuse is broken in individual rolls and returned to the cones with the run of mine feed. This is done in order to recover caps of coal that cling to egg and stove sizes.

Record of Cone Operation

The final slate from each of the four 15-ft. cone units is sampled at intervals of 12 min. and accumulated for each 1-hr. period. The four samples are taken to the testing room, where each sample is placed individually in the Delatester to obtain the percentage of float in the slate end from each unit. This work is performed by the plant control tester, whose specific duty it is to conduct these tests for each hour of the operating day. As soon as he has obtained the percentage of float, he visits the four cone operators and advises them of the results. Each cone operator marks the percentage of float in his record book.

These record books contain all data pertaining to cone operation, shown for each hour as follows:

1. Percentage of float in final slate.
2. Pounds of water pressure per square inch as recorded by pressure gage.
3. Number of turns on valve admitting water to top spray line, to center spray, to bottom spray line; to classifier column.
4. Delays to operation.
5. Time sand was added to system.

This information for an 8-hr. operating period would appear as in Table 1.

Table 1.—*Record of Cone Operation*

Hour	Water Pressure Lb. Per Sq. In.	Turns on Valve Admitting Water				Float Final Slate @ 1.65 Sp. Gr. Per Cent.	Remarks
		To Classifier Column	To Top Spray	To Center Spray	To Bottom Spray		
7:00– 8:00	20	2½	0	3	3	2.1	
8:00– 9:00	20	2½	0	2	3	1.0	
9:00–10:00	20	2½	0	2	3	0.76	
10:00–11:00	20	2½	0	3	3	1.2	
11:00–12:00	19	2½	0	3	4	1.1	Sand added
12:30– 1:30	19	2½	0	3	4	2.8	
1:30– 2:30	19	2½	0	2½	3	0.81	
2:30– 3:30	19	2½	0	3	3	1.50	
Average						1.41	

Following the basis of regulation, between 0.5 and 1.5 per cent., we note that the float in the first operating hour was 2.1 per cent., which is above the allowable loss, and that in the second hour this dropped to 1.0 per

cent. The change was caused by cutting down one turn on the valve of the center spray line. This setting was satisfactory for 2 hr., then the cone operator felt that the percentage of float was getting too low and added one turn on the center spray, which caused a slight increase in the percentage of float. A further change was made in the fifth hour, with the float remaining about the same. The cone operator misjudged slightly, as in the sixth hour the float jumped to 2.8 per cent. Immediately he cut off one and one-half turns of water, and as this brought the float down too low, he added one-half turn. The average float for the day was 1.41 per cent., which is about what we try to maintain. Our sales department is satisfied with the market product when the float in the final slate is in the neighborhood of 1.5 per cent.

Operation of Spray Lines

Table 1 indicates that only the bottom and center spray lines are used for supplying agitating water, as the water from these lines is more evenly distributed throughout the fluid mass than the water admitted through the top agitating spray line. For that reason the water is increased or cut down as necessary. The classifier-column remains constant throughout the day. In operation, only sufficient water is used at this point to hold the sand in suspension and still allow the slate to drop freely through the classifier column.

The cone operator also keeps a record of all delays that result in a stoppage of the feed. This information, together with plant control data, tell the operating officials at a glance the performance of each unit.

The hourly record as kept by the cone operator indicates spray blockage. Suppose that with a cone using a normal supply of water, as indicated by the setting of the valves in the form, the plant control tests report 0.5 per cent. float. The cone operator adds so many turns on the valves. The next hourly report shows zero float; the operator immediately adds more turns and keeps adding turns on the valves, but the float in the final slate does not increase. This indicates that the spray line is blocked. If the sprays cannot be cleaned by blowing the line free, this unit is emptied after operations, and invariably one-half or more of the sprays are found to be blocked. A unit operating in this fashion results in an uneven preparation. When such a condition occurs, the unit must be watched very closely until it is again in proper operating condition.

Roll Crushing Slate End

As mentioned before, two sets of rolls are employed to crush the slate ends; that is, the egg size is broken to stove and smaller in one set, and the stove size is broken in the other set to chestnut and smaller. The

material passing through each roll is returned to the cone with the run of mine feed. Experience has shown that when the float in the final slate goes above 1.5 per cent. there is undue degradation in these rolls. A typical loss from this course would be:

Float at 1.65 Sp. Gr., Per Cent.

Final Slate	Egg Roll	Stove Roll
1.5	4.0	4.0
2.2	8.3	5.3
3.8	11.0	13.0

From this it is calculated that when the final slate shows a loss of 1.5 per cent. the egg and stove sizes being broken in the refuse rolls contain 4 per cent. of float or recoverable material, and when the loss in the final slate increases, the loss in the roll material also increases. This results in undue degradation in addition to increased losses to the slate ends. It also increases the volume of material in recirculation. This is all controllable by the hourly plant control tests.

Control by Ammeter

At the cone operator's station there is an ammeter that registers the load on the cone. This meter is graduated in kilowatts from 0 to 30 in 1-point readings. When the load registers 10 kw. on the ammeter, the operator opens the top slate gate and traps out the slate. After completing the trap the operator does not work the slate gate until the ammeter again registers 10 kw. This method is employed throughout the day to regulate the removal of slate. The normal load is about 5 kilowatts.

To further facilitate cone operation, there have been installed at the cone operator's station ammeters that register the load on the 10 in. recirculating sand pump and the 4 in. make-up sand pump. These pumps are located two floors below the cone operator, but he can tell instantly by the ammeter whether the pumps are functioning properly. If either pump does not register 15 kw., or full load, the attendant in the vicinity of the pump is notified by means of a klaxon signal. He immediately corrects the condition.

By this method of control the operator can obtain a consistent performance throughout the entire operating day and the operating officials can know definitely the performance of each unit and can take steps to remedy any out of line condition immediately.

To properly apply this system of control, other important conditions must remain constant, or nearly so; *viz.*, feed regulation, sand loss and silt removal.

After the feed regulation had been carefully studied, a Ross feeder was installed, which operates very successfully. This feeder is simply a series

of revolving chains across the width of the chute, which allows only a certain volume of coal to pass over the shakers and thereby regulates the

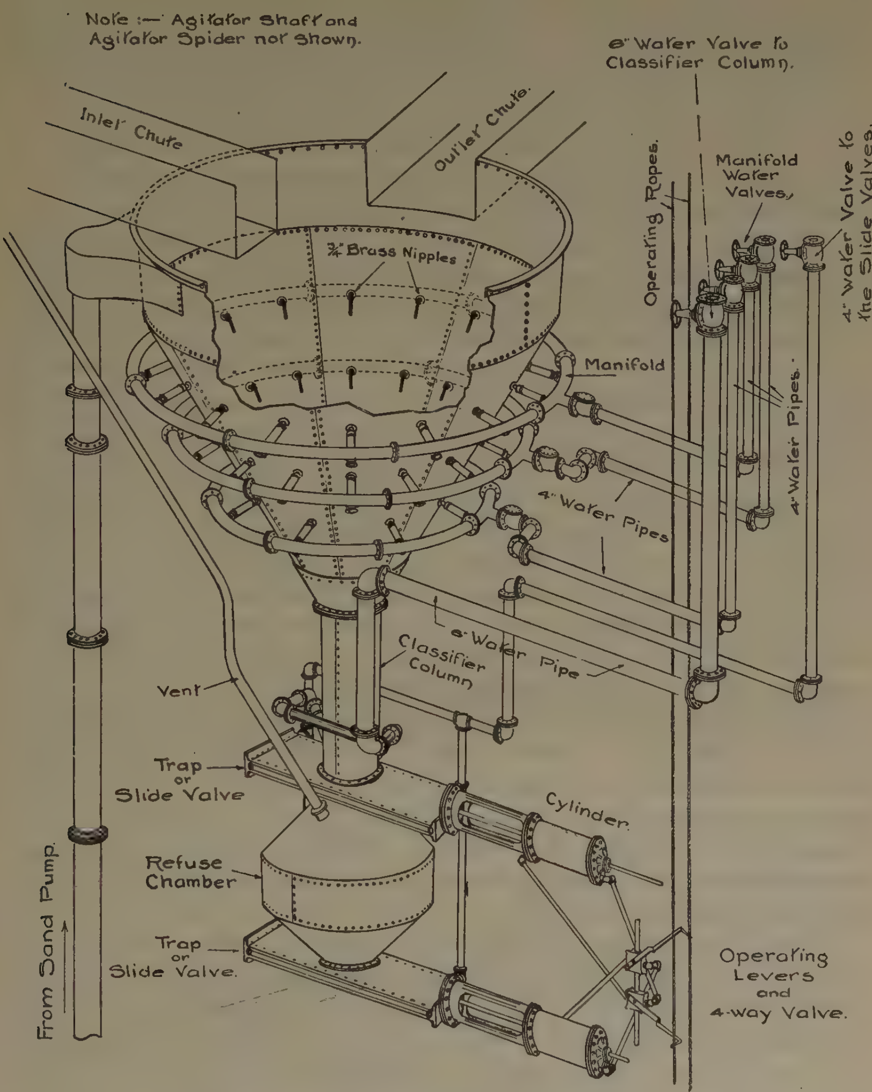

FIG. 1.—CONTROL SYSTEM FOR CHANCE CONE.

volume to the separating cones, which is estimated to be about 200 tons per cone per hour.

Sand losses are noted from the cone operator's record book. If the day's operating report shows that more than a normal amount of sand is

being used, steps are taken to find the cause and to remedy it immediately. Sand losses cause a low sand line in the separating cone, which does not allow the floating mass to pass out through the discharge and forces an excessive amount of material to be carried around the cone until eventually a floating island is formed. As this will not allow the slate particles to sink, it usually results in heavy slate in the market product.

Another cause of sand loss is the breaking of a shaker arm or some part of the shaker, which forces the domestic or desanding shakers to stop under load. If the 10-in. sand pump were allowed to run, there would be considerable sand loss. This loss has been controlled by installing an emergency stop at the cone operator's station. If, for any reason, the domestic shakers stop under load the cone operator immediately stops the 10-in. recirculating pump by pulling the emergency stop. The attendant in the vicinity of the sand pump starts the pump when given a signal to do so.

Another feature that has been given considerable study is removal of silt. This is accomplished on all shakers handling coal before the coal passes into the separating cone. Any silt that is not removed by the shakers, or in the travel of the coal from the shakers to the cone, is removed by a skimmer arrangement. At the Marvine Colliery of the Hudson Coal Co. this arrangement is located in the chute carrying the sand, coal and water from the cone discharge to the domestic shakers. The silt riding on top of the fluid mass is skimmed off and placed on the back end of the steam-size shakers, where the sand is removed from the silt and returned to the system. The silt passing over the $\frac{3}{64}$-in. jackets is sent to the silt plant. This silt runs about 10 per cent. ash.

At another colliery, the chute arrangement would not permit a duplicate of the Marvine installation. It was necessary to tap the separating cone and insert a 3-in. pipe, which extends 18 in. into the cone. This arrangement skims the silt as it is being carried by centrifugal force around the cone toward the discharge. The silt passes out of the cone through the 3-in. pipe and is placed on $\frac{3}{64}$-in. jackets, which reclaim the sand and discard the silt.

Tramp iron has caused difficulty in the operation of the refuse rolls by coming into the breaker in the feed and following the course of the slate into the refuse rolls. To overcome this, at one colliery magnetic pulleys have been installed at the head of the belt conveyors over which the slate passes to the rolls, and at Marvine magnets have been installed directly above the slate shakers. The records at Marvine show 907 lb. of tramp iron removed over a period of one week from a single unit.

At Marvine the refuse from the separating cone discharges into a scraper line; at the other colliery, there is free slate discharge, which the company believes is a better arrangement. The free slate discharge also renders unnecessary the maintenance of the scraper line and booth arrangement used at Marvine.

Performance of Cones

As a means of checking the performance of the cones by plant control tests, it is compared with the bank tests made by the Inspection Department. The bank test consists of sampling the face of the slate bank and hand-picking the material to obtain the percentage of coal and the percentage of bone lost with the slate from the cones. A typical comparison of these results would be as follows:

Size	Sample Slate Bank, Lb.	Hand-picked Analysis, Per Cent.		Float @ 1.65 Zinc Chloride, Per Cent.	
		Coal	Bone	Coal	Bone
Chestnut	335	2.0	2.75	1.00	0.37
Pea	40	1.8		0.13	
Buckwheat	24	2.5		0.06	

This indicates that the losses as shown by the plant control tests compare favorably with visual inspection.

Since the operation of the Chance cones at Marvine, the following performance as to condemnation has been obtained:

Number R. R. cars loaded	73,860
Number R. R. cars condemned	3,793
Percentage condemned	5.1

Of the total condemnation of 5.1 per cent., only 0.2 per cent. is chargeable to Chance cone operation.

DISCUSSION

(*Paul Sterling presiding*)

J. R. Campbell, Pittsburgh, Pa.—Does Mr. McLaughlin measure the quantity and quality of the sink material in the washed coal?

J. F. McLaughlin.—No, we do not. We are guided on our washed coal by the inspector who is at every operation. The divisional analysis he gives every car is our guide. We find, in practice, that the 1.65 sp. gr. that we maintain will satisfy the sales department as far as the washed product is concerned.

J. R. Campbell.—What is the quantity of sink material?

J. F. McLaughlin.—I would say it is about 1.5 per cent. at 1.65.

B. H. Stockett, Minersville, Pa.—What is the sand consumption per ton of prepared coal?

J. F. McLaughlin.—Our sand consumption is approximately 4 to 5 lb. per ton shipment. Our last operation, which was converted from jigs to Chance cones, is slightly better than that. The last time I figured sand loss, which was about one month ago, at this operation, it was not quite 3 pounds.

B. H. Stockett.—On that basis, what is the percentage of refuse in the incoming feed?

J. F. McLaughlin.—The incoming refuse is increased because the egg and stove coal is rebroken and returned. The final rejection amounts to approximately 12.5 per cent. The incoming feed probably will be 20 or 22 per cent.

W. H. Lesser, Frackville, Pa.—What size coal do you put into your cone?

J. F. McLaughlin.—Rice coal—above $1\frac{1}{64}$-in. mesh—up to egg coal; that is, over $3\frac{1}{4}$ in. and through $4\frac{6}{16}$.

J. R. Campbell.—Mr. McLaughlin tells us that the market products are based on visual inspection. Is the refuse or bank material based on the sink and float?

J. F. McLaughlin.—The market product is subject to two inspections—visual analysis and hand-picked analysis—and it is again subjected to visual analysis.

J. R. Campbell.—And is the refuse material passed on by the sink and float control entirely?

J. F. McLaughlin.—No, it is based on the Standard Conference limits of slate and bone allowed.

C. Evans, Jr., Scranton, Pa.—Mr. McLaughlin refers to certain standards of allowable impurities set up by the Anthracite Institute. As a matter of accuracy in this discussion, I would say that the present market conditions of anthracite are such that we cannot ship coal that has the allowable limit; so that the actual shipped product from the Chance cones carries much less than the allowable limit as set by the Institute. That is true of every other anthracite-producing company.

J. R. Campbell.—It would be interesting if you would give some of the tolerances on the different sizes.

C. Evans, Jr.—Do you mean the theoretical or the actual tolerances?

J. R. Campbell.—I mean the actual tolerances.

C. Evans, Jr.—I cannot give the actual tolerances because they are based on visual analysis, but in egg coal, for instance, the Anthracite Institute allows 2 per cent. slate and 3 per cent. bone. That is what they used to allow in the good old days. It allows 3 per cent. slate and 4 per cent. bone in stove coal, and in nut coal 4 per cent. slate and 5 per cent. bone. But in the actual market conditions today we cannot sell coal containing that amount of impurity, and for our actual impurities I will have to refer you to Mr. McLaughlin.

J. F. McLaughlin.—I would say that it is common for us to produce a car of egg coal or stove coal with zero percentage of slate and 1.5 per cent. bone; on chestnut 1 per cent. slate and 1 per cent. bone; on pea coal 4 per cent. slate and on buckwheat from 2.5 to 3 per cent. slate. Generally egg coal will run 0.5 per cent. slate, and stove coal 1 per cent. slate. The so-called slate is not exactly slate; it is more bone that is classified as slate because it may be a little heavy. It may have a little cap.

B. H. Stockett.—How do you find the Chance cone plant compared to the jig plant as far as condemned coal is concerned?

J. F. McLaughlin.—Our trouble with the jig plant today is in trying to remove the intermediate product. Our jigs cannot do that without sacrificing the bank. A

jig will do all it is expected to do; it will remove the heavy slate, and in the good old days when 2 per cent. slate in egg coal and 3 to 4 per cent. slate in stove coal could be shipped the jig was a good machine, but with present market conditions, requiring zero percentage of slate, or 1 per cent. slate, a jig cannot be used, so that there is really no comparison between a Chance cone and a jig.

J. R. Campbell.—How does that affect the prepared yield?

J. F. McLaughlin.—On our prepared yield the loss on jig operation from increased losses from reject to the slate bank and improvement from Chance cone preparation by decreasing the slate in the market product just about balance one another. In other words, the lower amount of slate just about balances the increased loss of rejects.

J. Griffen, Pittsburgh, Pa.—Mr. McLaughlin, have you ever run a jig plant that ground the reject from the primary jig to a smaller size, and re-treated that product in another jig? I think that the Hudson Coal Co. has really met the preparation problem, in that it has acknowledged that a specific gravity separation and a separation that meets visual inspection do not always agree. It has faced that problem and made the separation on a gravity basis. Because there is marketable material in the egg and stove and broken sizes in the primary refuse, it has crushed that product and re-treated it, which is the logical thing to do, but does it not also involve a reduced prepared yield unless that loss, as Mr. McLaughlin points out, is kept down to an absolute minimum?

In the material described by the table on page 328, is there not additional marketable material going to those rolls, which is heavier than 1.65 sp. gr.; that is, material which, on a visual classification, would be accepted by the coal inspector if it were in the railroad car?

J. F. McLaughlin.—Answering the last question first, I would say that the material above 1.65 is not marketable.

J. Griffen.—None of it? Yet the table on page 331 shows that chestnut size in the final refuse has twice as much marketable coal as that which floats at 1.65 sp. gr., while pea size has over 10 times as much.

J. F. McLaughlin.—None of it. There are certain pieces that have a slight cap. Ninety-nine per cent. of it is not marketable. For that reason, we make our separation at 1.65 sink and float. If the normal loss, which we try to maintain, of between 0.5 per cent. and 1.5 per cent., were jumped to 2.1 or 2.8, the table on page 328 would not represent what has actually happened, because the regrinds of the egg and stove coal would not be as shown in the table. It would increase in greater proportions. The reason it does not increase here is because of the additional refractures the material gets. It comes back and is reclaimed in egg, stove, or general line sizes—chestnut sizes.

As regards the yield on Chance cones and jigs, we know from experience in running actual tests of a day's output, even in view of the smaller amount of slate being commercialized, the yield from our Chance cones is slightly better than our jig operations.

A jig cannot break down the refuse, as the Chance cones do, because a jig today in the anthracite field has all and more than it can do to separate the existing impurities. As I said earlier, it is not the refuse that we object to from the jigging standpoint; it is the intermediate product from 1.60 to 1.70 sp. gr. that we want to get out of the washed coal, which we cannot do with a jig because it is lighter and the jig has a tendency to throw it with the coal.

If you try to operate the jig to get that material down to the slate, the jig will run on a light slate bottom, and anything is liable to happen. The jig may be tipped

and the slate bottom lost. When the operator is building it up again, if he notices that it is coming up into the coal end, he begins to lessen it, and then he notices that he is not getting enough out. He is adjusting that jig frequently, which causes inconsistent operation. The Chance cone eliminates that human element with the sink and float test on the refuse.

We have four cones at Marvine. We have one plant-control man, who is the brains of the four cone operators. He tells them what to do, and we know whether he is doing his part or not by the trend of the performance for the day. We know positively from our bank loss, as determined from inspection department tests and from our own zinc chloride test, that this system is operating successfully. They check up well and over a period of 1½ years we have not noticed any uneven trend.

B. H. Stockett.—In further answer to Mr. Griffen's question, we have two plants adjoining, handling the same seam of coal. On the Chance plant we operate on a fairly uniform feed. We do jig our refuse from the Chance plant in order to get the half and half material and to break it down again. In the jig plant, in order to meet market conditions, we jig the coal and rejig the coal end. The refuse is again treated and rejigged, and the coal end of the rejig is cracked down and put back into the preparation.

Even on that basis, while our tests have not been particularly elaborate, we find that on a sink and float there is about twice as much material on the float going out in the bank on the jig plant as there is on the Chance cone plant. We believe that with a Chance installation we would split about fifty-fifty on our car yields, and just hold the car yield and still have a better grade of coal for the market.

J. Griffen.—You feel that you get improved prepared yield because you save breakage and continual rejigging?

B. H. Stockett.—We are not even figuring on that because the prepared in the one mine is practically the same as the prepared in the other. We feel that we just about hold our own, but we still have a better product and a lower ash product to ship to the market than we have with the jig-operated breaker.

J. F. McLaughlin.—We find that the Chance yield is just a little better than the jig, but it is not a question of yield today, it is a question of marketable product.

W. H. Lesser.—It is not necessary to operate the Chance cones as Mr. McLaughlin describes. We have four cones in operation and we find that it is not necessary to use the sprays. The sprays are not touched during an entire day. The valve that supplies the classifier column with water is set in the morning, and unless something happens to the operation of the cone, that also is not changed. We do not use the gravity balls. Our operators use a stick that they put into the cone, thereby feeling the consistency of the fluid mass. These indications tell the operators whether they should speed up the trapping rate or reduce it. We control the operation of the cone by the inspector's report. The inspector is not far away from the breaker, and as soon as he finds slate coming out of the cone, he immediately notifies the operators. Since we have no pockets in the breaker, there is very little time between the inspector's discovery of poor separation and the operator's notification of it. Possibly one poor car of coal would be made during this period.

The Hudson Coal Co. has a remarkable sand consumption—4 lb. per ton. We do not nearly attain that figure. However, we put smaller coal into the cones than Mr. McLaughlin speaks of. We prepare in a single cone the entire product, broken coal to +3⁄32-in. mesh. When we first laid out our breaker, we planned to prepare pea coal and under in one cone and nut and over in the other, but we find a complete mixture to be better.

As far as condemnation is concerned, in the year 1929 our condemnation was less than 0.5 per cent. We shipped during that year 711,203 tons, and the condemnation was 3030 tons of coal; and of the coal that was condemned, 720 tons was condemned on account of sand, 330 tons on account of refuse and 1535 tons because of sizing. We have strippings from which we get some stained coal, of which 265 tons was condemned. Ashes were 180 tons. These figures answer Mr. Griffen's question.

We ship egg and stove coal consistently without any slate. The slate in the other sizes is far under the limits allowed by the producing companies.

Last year the average of all the tests made of the refuse bank shows that it contained 1.75 per cent. of coal, including the chipped coal. (Chipped coal is the coal that is chipped from a piece that is part slate and part coal.)

J. F. McLaughlin.—The reason that you are not using your spray is because you are maintaining a 1.72 sp. gr. We cannot afford to do that. There is too much intermediate in our feed for us to maintain that; if we used all of our water in the classifier column, it would have a tendency to hold the slate in suspension. The classifier column water can only be regulated in the point where it holds the sand in suspension and still allows slate to drop free.

We operate the slide valves on an ammeter. The cone operator does not touch the slide valve until the ammeter registers at a certain point. Then he traps out. We use the same system of the stick, but not to test the gravity, because it is impossible to do it in that fashion. The stick is used to find out whether there is a low float.

As regards sand loss, we introduced facilities at Marvine, and the gravity slope, which is 18-ft. settling cones. Another thing that you have that we have not is that fine coal above $\frac{3}{32}$ in. goes in. It is the same thing as when silt accumulates in the top section—as it passes on to the shaker, the silt has a tendency to carry forward and carry sand with it. Probably that is the cause of your sand loss. We have not that condition because we grade our sand loss. We try to break it off at least in the third descending jacket.

C. Evans, Jr.—Mr. Lesser, who has charge of your inspectors? Are they under the sales department or the operating department?

W. H. Lesser.—The inspection of our coal comes under the supervision of the general manager.

H. G. Turner, Bethlehem, Pa.—Have minerals other than quartz and sand been tried in the Chance cone?

J. F. McLaughlin.—Trap rock is the only thing I know of. Trap rock and silica sand are about the only two we need. Even with silica sand, we have to use about 700 gal. of water per minute to cut it down to the required 1.65 sp. gr. solution, so that with a mineral heavier than that we would have to use twice or three times the volume of water, and the water is not available. We have to recirculate the water now in order to get through.

C. Evans, Jr.—Answering that same question, we made another experiment with which Mr. McLaughlin is not familiar, in which we attempted to use pulverized waste from iron ore taken from our Chateaugay iron-ore mines. The material was so heavy we could not handle it successfully.

Heat Drying of Washed Coal

S. M. Parmley,* Pittsburgh, Pa.

(Pittsburgh Meeting, September, 1930)

Experience has shown that there are some factors connected with the drying of fine washed coal that are not present in drying of slack coal as normally practiced at cement kilns or pulverized coal plants. It is the purpose of this paper to show some of these variations as they have been encountered under actual operation.

The Champion No. 1 cleaning plant of the Pittsburgh Coal Co. uses the wet system of cleaning coal. The plant is equipped to produce commercial sizes or mixtures, the primary sizes being plus 6 in. and 4 by 6 in., which are hand-picked. Minus 4-in. sizes are all cleaned by the wet process.

To produce a commercial product as free of moisture as possible, the washed coal is first passed over sizing and dewatering screens, then each size except the $-\frac{3}{8}$-in. coal is conveyed to loading booms by slow-moving draining conveyors. As the $+\frac{3}{8}$-in. sizes are not heat-dried, no data will be given, as it is the purpose of this paper to describe the heat drying of $-\frac{3}{8}$-in. coal as practiced at the Champion No. 1 plant of the Pittsburgh Coal Company.

To fully appreciate the conditions, it must be remembered that the $-\frac{3}{8}$-in. product is now shipped at a lower moisture content than when received from the mine. The incoming coal in the $-\frac{3}{8}$-in. size will average 5 per cent. moisture and the shipped product 3 per cent. moisture. This is a case where a wet process gives a reduction of both ash and moisture in the fine sizes. The shipped product, therefore, is more uniform in all characteristics, and uniformity of product is the most important single factor in coal preparation.

The 0 by $\frac{3}{8}$-in. coal is first dewatered on slow-moving continuous bucket dewatering elevators, running at the rate of 40 ft. per minute, with 48-in. perforated buckets with $\frac{1}{4}$ by $1\frac{1}{2}$-in. slots, spaced $\frac{3}{8}$ by $2\frac{1}{8}$-in. centers staggered. The moisture in the coal discharged from the elevators runs 20 to 28 per cent., depending on the fineness of the product. The discharge from the dewatering elevator is further drained on a drag-type conveyor equipped with $\frac{1}{2}$-mm. wedge wire screen.

The conveyor discharges coal with an 18 to 25 per cent. moisture to three centrifugal dryers of the Carpenter type. The Carpenter dryers

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discharge a product at 6.5 to 8.5 per cent. moisture on to a drag conveyor, which in turn delivers to three 50-ton storage bins in the dryer plant. Variation in moisture from dryers depends on the character of feed, especially as regards percentage of −100-mesh material.

The discharge from the Carpenter dryers runs from 100 to 120 tons per hour. To this is added 20 to 25 tons per hour of filtered product averaging 21 per cent. moisture. Oliver, Dorr and Laughlin type filters are used. A composite of the products from the centrifugal dryers and the filters form a feed to the heat dryers. Table 1 gives data relative to coal feed to three heat dryers at Champion No. 1.

TABLE 1.—*Coal Fed to Three Heat Dryers*

	Centrifugal Dryer Product 0 by 3/8 In.	Filter Product	Composite Feed to Heat Dryer	Discharge from Three Cyclones
Tons per hour	100 to 120	20 to 25	120 to 145	3
Moisture in feed, per cent	6.5 to 8.5	21	7.5 to 9.5	
Moisture in discharge, per cent			3.0	1.5
Screen Test, Per Cent. Weight				
On 3/8 in. round	0.0	0.0	0.0	0.0
14 mesh[a]	72.0	0.0	62.0	20.0
48 mesh	20.5	33.0	22.0	48.0
100 mesh	3.5	31.0	7.5	22.0
200 mesh	1.5	14.0	3.5	6.0
−200 mesh	2.5	22.0	5.0	4.0
	100.0	100.0	100.0	100.0

[a] Tyler standard sieves.

No mixing arrangement has been provided for mixing the products from the centrifugal dryers and the filters except that obtained by passing through the conveyors and storage bins. It is the opinion of the writer that the mechanical handling through the heat dryers and drying of the finer coal would be much improved by providing some mechanical means of thoroughly mixing the two products before entering the dryer.

From each storage bin the coal is fed to each heat dryer by a flight conveyor feeder moving 20 ft. per minute with low flights 24 by 1 by 2 in. high. Feed is regulated by a vertical rack and pinion gate in front of bin. The feed conveyor discharges into a special feed spout and air-discharge hood on dryer. The dry coal from the dryers discharges on to a common flight conveyor, which discharges on to the return side of the wet-coal conveyor that carries coal to heat-dryer plant. This conveyor carries the coal back to the loading shed of the cleaning plant, where it may be loaded separately or as a mixture with the larger sizes.

The mechanical handling of wet 0 by ⅜-in. coal is rather a problem. To prevent hanging or arching in the bin, all bins were made with 60° sloping bottom and vertical ends. The bottom is open, leaving a long narrow opening 19 in. by 10 ft. 6 in. The bin was later equipped with sliding gates 18 in. wide to take weight from the chain. The required number of gates are opened to prevent arching in the bin and keep a steady flow. As an additional precaution, poke holes were placed on three sides of the bin.

The fine wet coal tends to build up on the conveyor bottom and raise flights and chain. To prevent this, flights were made with serrated teeth, which keep the conveyor bottom clean. As considerable material sticks to flights, provision was made on the return side of the conveyor to take care of this spillage. At Champion No. 1 the material is carried to a spout leading to the stoker feed hopper. Flight conveyors have been used throughout for handling the dry coal, as such conveyors stand the heat from the coal and are more readily enclosed and made dust-tight. Conveyors may then be equipped with stacks for carrying off surplus heat and moisture.

At Champion No. 1 there are three dryers, 102-in. dia. by 65 ft. long, type BD, as manufactured by L. R. Christie Co., Pittsburgh. Table 2 gives data relative to the dryer.

Table 2.—*Data on Dryer at Champion No. 1 Plant*

Type	BD
Size	102 in. by 65 ft.
Weight	130,000 lb.
Shell plate	9⁄16 in.
Lifting flights	15 in. deep by 7⁄16 in. plate
Inclination	⅝ in. per ft.
Size motor	60 hp.
Speed of drum	5 r.p.m.
Gear ratio	12 to 1
Brick per dryer—with stoker setting:	
Common brick	12,000
9 in. straight first quality firebrick	8,000
No. 1 arch first quality firebrick	500
No. 1 wedge first quality firebrick	150
Capacity per dryer	45 tons per hr.
Size coal	0 by ⅜ in. (See sizing test.)
Moisture feed	9.5 per cent.
Moisture discharge	3.0 per cent.
Inlet temperature of gases	1300° to 1400° F.
Outlet temperature of gases	130° to 150° F.
Discharge hood, temperature of gases	800° to 850° F.
Coal burned per ton coal dried	15 lb.
Heating value coal, dry basis	14,100 B.t.u.

Feed to Dryers

With fine coal as handled at Champion No. 1 there was considerable trouble with coal building up in the feed spout to the dryer. This was overcome by using a spout with a half-round section, removing all obstructions in spout, making surface smooth, and increasing the slope from 45° to 70°. To overcome rusting action, spouting was made of Ascoloy steel. The wet coal would not get away from the feed end of the dryer fast enough to prevent a backing up and leak through sealing rings. This was partly overcome by placing additional flights at the feed end, laid across the axis of the dryer.

The mechanical conveying capacity of coal through the dryer was reduced by a build-up of fine coal caking on lifting flights. At first automatic hammers were used on the outside of the shell to loosen the caking on the flights. The hammer rested on the shell and tripped periodically on lugs. This arrangement required so much maintenance that it was discarded. At present, the dryer is emptied about once a week and pieces of scrap iron are thrown in and allowed to pass through the dryer while revolving. This system is working well.

Various speeds were tried, from 2 to 5 r.p.m. More complete drying and a larger capacity was obtained at 5 r.p.m., at which the dryers are run. Although material passed through the dryer much faster at higher speed than at lower, the more complete drying effect is due to the action of the air on the thin stream of coal passing through the dryer at the higher speed. At the lower speed the flights were completely filled and a large bulk of the material rolled over on itself on the bottom of the dryer; at faster speed the flights were only partly filled and the material was carried up to the top of the dryer and allowed to spill through the gas stream. At 5 r.p.m. it takes 20 min. for material to pass through the dryer.

The inclination of the dryer is ⅝ inch per foot. This is fixed by the manufacturer, so no tests have been made as to the effect that variation in inclination would have on performance.

Heating Dryers

Each dryer is heated by an inclined-grate overfeed stoker, 8 ft. wide by 5½ ft. long, made by the Cokal Stoker Co. The stoker is the automatic variable speed type, equipped with steam or air jets over the fire for burning fine coal. It has a No. 40 type SP Clarage blower and damper regulator for operation with forced draft. The blowers are used only when dryers are being started, in order to get the heat up quickly. As soon as the desired temperature is reached, the blowers are stopped. The induced-draft fans on the dryers produce enough draft through the fire to give the required heat regulation.

The "over-fire" jets are not used, because they force sparks into the dryer as far back as the discharge end, thereby causing a possible fire hazard. The jets give more complete combustion and keep feed hopper and pusher bars cool. As soon as the jets were eliminated, the fire carried back into the feed hopper and on to the retort housing. As a result, the pusher bars and retort housing burned off and caused a high maintenance cost. This was largely eliminated by opening the front firing doors of the stoker as well as the ash-pit doors. This over-fire air not only cooled the feed hopper but increased the drying effect of the dryer by allowing more air to pass through the fan for a given speed. The ash-pit doors are always open except when the blower fan is being used, in order to keep the grate bars cool and prevent them from burning out.

The stoker is fired with 0 by ⅜ in. wet coal from storage hopper to dryers and has the same moisture content (9.5 per cent.) as the feed to the dryers. The stoker is designed to burn 1200 lb. of 0 by ⅜ in. wet coal per hour. Tests show that only an average of 675 lb. of dry coal per hour is being burned, for a capacity of 45 tons per hour of dry coal feed to each dryer, with a moisture content of 9.5 per cent. in feed, drying down to 3 per cent. moisture in discharge from the dryers. This gives 15.0 lb. of dry coal fired per ton of dry coal dried, or 9.8 lb. of water eliminated per pound of dry coal fired. Moistures are samples oven-dried for 12 hr. at 85° C. The fuel has a heat value of 14,100 B.t.u., dry basis. The ashes are removed manually by operators.

The furnace is divided into three parts—the stoker furnace, combustion chamber and distributing chamber. Fig. 1 shows its design and dimensions. The furnace walls are laid with 13½ in. of common and 4½ in. of fire brick. The furnace and combustion chamber are covered with a Detrick suspended arch of flat construction. A distributing arch of checkered brick is placed under the dryer shell in the distributing chamber. The arch was originally made with a double row lock of arch brick 9 in. high, but this soon collapsed. The arch was then replaced with a specially shaped brick 9 in. high. The furnace is equipped with clean-out doors and inspection holes.

An asbestos-block sealing ring held in place by an iron-band expansion hoop is used to prevent air leakage between the dryer shell and brickwork.

An auxiliary flue and damper are provided on each furnace, leading to one main stack. When feed is stopped on the dryer, or in case of an emergency shut down, gases are by-passed directly to the auxiliary flue and stack.

Each dryer is equipped with indicating pyrometers placed in front of the furnace convenient to the operator. One pyrometer indicates the temperature of gases entering ports of the dryer; the other indicates the temperature as the gases leave the inner tube of the dryer at the

discharge end. Each pyrometer has a high-temperature and a low-temperature alarm.

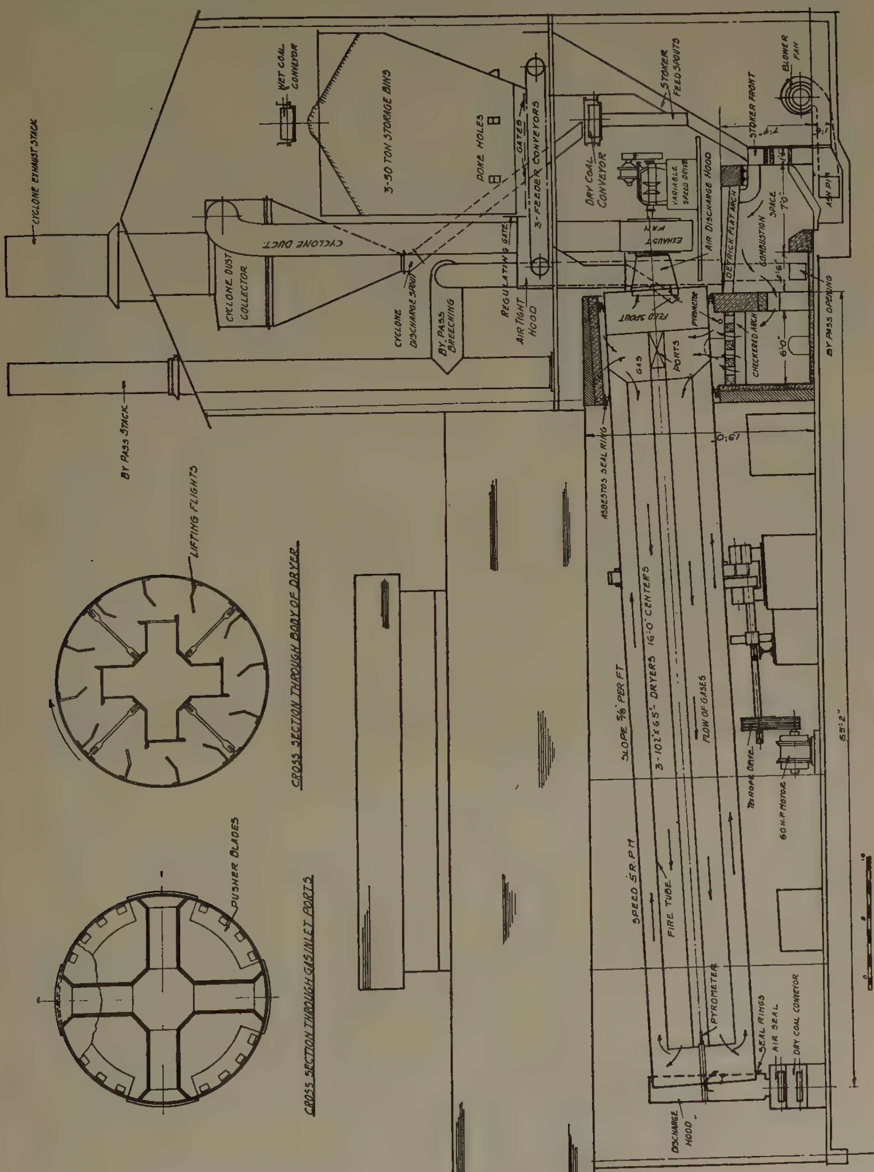

FIG. 1.—DETAILS OF FURNACE.

In the superintendent's office there is a 6-point recording pyrometer, giving temperature of inlet and outlet gases of each dryer on one chart. The temperature of inlet gases to dryer is held between 1300° and 1400° F. The gases at discharge end are held between 800° and 850° F. The

temperature of gases entering the induced-draft fan at the fan inlet or discharge port of dryer is 130° to 150°, depending on the inlet temperature of gases. This temperature of gases is held for normal running of the dryer; that is, for a tonnage of 45 tons of dry coal, with screen test as given in Table 1 and a 6.5 per cent. moisture reduction of coal.

A change in the moisture content and fineness of the feed reacts within 5 to 10 min. on the moisture in the coal from dryer discharge. The operator then changes the air circulation, which reflects back on the temperature of the furnace and dryer. No automatic system has been attempted as yet on regulation of temperature and air in order to maintain a constant moisture content of discharged coal. At present such regulation is made acccording to the judgment of the operator, who regularly inspects the coal as it discharges from the dryer.

Fans and Cyclones

Each dryer is equipped with a Buffalo Forge Co. No. 80 Type M standard mill exhauster, driven by a Llewellen variable-speed transmission and a 15-hp. motor. A manual control of the transmission is placed in front of the furnace, accessible to the operator.

The inlet of the fan is connected directly to the feed hood of the dryer. The fan discharges through a 28-in dia. duct to a cyclone dust collector of the spiral type, one for each dryer. The cyclone collector is 10½ by 16 ft. high, made of 3⁄16-in. plate, heavy enough to withstand any corrosion due to condensation, or a build-up in the cyclone collector due to damp coal.

The cyclone discharges through a spout to the dry-coal conveyor or to the stoker feed hopper. The exhaust gases and moisture-laden air pass to the outside of the building through a stack (60-in. dia. by 15 ft. 7 in.) on top of each cyclone. The top of the stack is 66 ft. above the furnace floor.

The cyclone is placed inside the building to prevent its being exposed to cold outside air. This gives practically no condensation or build-up of damp coal on the sides of the cyclone. Some condensation takes place, however, in the stack from the cyclone, which drops back into the cone of the cyclone and helps to dampen the coal and cut down the dust problem in cyclone discharge. In cold weather the cyclone discharges steam rather than the expected dust cloud. Operating the dryer at 3 per cent. discharge moisture, there is no dust from the cyclone exhaust except during the period of closing down the dryer.

The air through the dryer is regulated by the fan speed, which is varied according to moisture content of the coal from the dryer discharge. The fan performance under normal conditions is shown in Table 3.

TABLE 3.—*Fan Performance under Normal Conditions*

Type	Standard PMX exhausters
Size	80 in.
Speed (variable)	450 to 600 r.p.m.
Capacity	10,000 to 17,000 cu. ft. per min.
Motor size	15 hp.
Inlet diameter	32 in.
Outlet diameter	28 in.

It is the opinion of the writer that the air regulation is far more important than the heat regulation. In the design of a heat dryer, more thought should be given to the proper proportioning of the air passages rather than to the use of high heats; in other words, using a large quantity of air at low heat rather than a small quantity of air at high heat. The high heats give a high maintenance on brickwork and produce too high a heat at the discharge end, which gives too great a fire hazard.

The present air hood or fan inlet at the feed end of the dryer is too small. As a result there is such a high air velocity that coal is pulled back into the hood and fan housing. This has to be cleared four or five times a day, otherwise the air passage would be blocked. A large amount of this coal is drawn into the fan and cyclones, which gives considerably more coal from the cyclones than under proper conditions.

The water gage at various points through the dryer is as follows:

Place	Inches
Over stoker furnace	0.3
Over checkered arch	0.5
End of dryer discharge hood	0.7
Inlet to air hood	1.0
Discharge of air hood or inlet to fan	2.25
Outlet of fan	0.75 due to draft effect of cyclone stack.

Fan speed, 585 r.p.m. Temperature of air at fan inlet, 130°. Air per minute to fan, 15,000 cu. ft. at 130° F.

The increase in temperature in the dryer varies directly with the quantity or velocity of air passing through the dryer. No attempt has been made to strike a heat balance or work out the thermal efficiency of the dryers, as there are so many variables to take into consideration and not enough systematic testing has been carried out to prove the figures.

The seals between dryer discharge hood, feed hood and dryer shell are made by means of an asbestos board bearing against the dryer shell. Due to expansion and movement of the shell, the asbestos packing soon breaks out and allows considerable space for cold-air leakage into the dryer. It is important for proper and efficient working of the dryer to prevent any air leakage through the feed spout, which would short circuit direct to the fan, also into the discharge spout from the dryer discharge hood. It is the opinion of the writer that if all the air could be taken in at the furnace a much smaller fan could be used.

Prevention of Condensation and Dust

The temperature of the coal as it leaves the dryer depends on the amount of moisture in the coal. At 1 per cent. moisture the maximum temperature recorded was 220° F., and at 3 per cent. moisture the maximum temperature recorded was 160° F. The temperature of coal in cars taken several hours after loading showed an average of 125° F., a maximum of 145° and a minimum of 95°.

All conveyors are enclosed and placed inside the dryer house. Each conveyor is vented at a high point with stack running to the outside of the building. A heavy condensation takes place inside conveyors if they are exposed to outside air. The moisture in the coal is increased, which soon causes a build-up on chain and troughing and not only gives mechanical trouble with the conveyor but also retards spouting of the coal.

Storage bins for dry coal are completely enclosed from outside air. Double wall insulation is used and in winter heat is placed around bins. The tops of the bins are vented to carry off saturated air and vapors. If bins are exposed to outside air, condensation takes place on the bin walls and not only increases moisture in the coal but forms a sticky mass on the bin bottoms that will not clear itself, and as a result the bins build up so that coal will not run out at all. Bins are circular with conical bottoms sloping 45°, and collapsible vertical discharge chutes of 30 in. diameter.

All dry-coal conveyors, elevators and spouts are enclosed and made dust-tight and vented so that there is a tendency to pull air into the conveyor, therefore plants are remarkably free of dust.

With a coal feed with a screen and moisture analysis as given under Table 1, "a mixture of filter product and centrifugal dryer product," the heat dryer has a capacity of 45 tons of dry coal per hour.

The operation of the dryer depends upon the fineness of the coal in the feed, also on the moisture in the dryer discharge.

The capacity of the dryer depends on two things: (1) conveying capacity or ability of coal to pass through the dryer; (2) air and heat conditions to dry coal to desired moisture.

For a given moisture content the dryer will convey the coarser coal much faster than a finer coal, or a coal with a higher percentage of −100-mesh material. The finer sizes seem to separate and adhere to the outside shell and lifting flights for about one-third the length of the shell from the feed end. The lifting flights soon fill and the coal rolls over on itself instead of being lifted by the flights. As soon as this condition exists, coal backs up at the feed spout and spills through the sealing rings.

If air and heat conditions are regulated so that moisture in discharge coal is much above 4 per cent., the capacity of the dryer is cut down

because of the inability of the dryer to convey the coal, as stated above. This is true for coarse as well as fine coal.

The maximum capacity of the dryer is obtained by operating with a coarse coal and drying to an extremely low moisture content; that is, operating within limits of air and temperature regulations.

The increased amount of fines in feed coal due to adhering to flights no doubt also increases the fire hazard.

As operated at Champion No. 1, the moisture in the product from the heat dryer depends on the requirements of the customer. The most desirable moisture seems to be about 3 per cent. Coal at 2.0 to 2.5 per cent. moisture is extremely dusty and is not desired by most customers on account of the dust problem while unloading. Coal above 5.0 per cent. moisture may give trouble in handling on account of damp fine coal building up in bins and chutes and retarding flow through the spouting, thereby giving an intermittent flow.

Tests run on coals at discharge temperatures as operated show that there is no appreciable change in volatile matter in coal after it has passed through the dryer.

Tests show that there is no measurable breakage through the heat dryer. Screen analyses show a slight increase in the +14-mesh material in discharge from dryers over that in feed. This is due to elimination of part of the 48-mesh material through the cyclone collectors.

Fire Hazards

During the first four months of operation there were several fires, due principally to lack of experience in operation, but since a systematic method of operation and a rigid set of rules were adopted, no fires have occurred in the dryers. Maintaining too high a temperature of air circulating through the dryer while stopped, or at a time when no coal was passing through, also drying coal at too low moisture with too high a heat at the discharge end of the dryer, seemed to be the principal causes of all the fires.

As a fire protection, the dryer is equipped with a water system with a connection at each end of the dryer. Valves are placed in front of the furnaces, accessible to the operator. In case of fire, shells are flooded with water and kept revolving. Fans are run at slow speed to keep the shells clear of gases and prevent gas explosions. Each floor is equipped with fire hose and nozzle, connected to the fire line. Two-gallon chemical extinguishers are placed at convenient locations for use in small fires.

Instructions for Operating Heat Dryers

The following set of operating instructions have been adopted at the Champion No. 1 plant.

The dryer feed bins should be left partly filled when shut down at the close of shift. Leave about 20 tons in each feed bin, or enough to run dryers about 20 min. before wet coal starts from the washery.

The dryer plant should start 20 to 25 min. before the washery plant, so that dry coal from the drying plant will arrive at the car at about the same time as washed products from the washery.

Coordinate time of cutting feed on the dryers with feed on the washery, in order to hold about 20 tons in each dryer bin at the close of shift.

Starting.—Start the dryer revolving about 15 min. before bringing up the heat, in order to free it of any dusty coal. See that the dryer is practically free of coal, close the stack damper, then bring the heat at the feed end up to 800° F. by starting the suction fan at low speed. Cut in feed and bring up heat to 1200° F. by gradually bringing up the suction fan to proper speed, keeping the outlet temperature to a point below 850° F. As discharge-end temperature drops, speed up the fan. Coal feed should be started at ¾ normal feed for about 20 min. until coal and dryer are heated, in order to prevent plugging at the feed end of the dryer.

Stopping:

1. Stop feed.
2. Stop stoker and reduce speed of suction fan.
3. Five minutes after cutting feed (or such interval as found by experiment), stop suction fan and allow gases to pass through coal and cyclone by natural draft. Cool off furnace gradually.
4. Twenty minutes after stopping feed (or such time as bulk of coal has passed through dryer), open by-pass damper and by-pass gases to stack.
5. Start to bank fire and by-pass gases to stack.
6. Revolve drum until all coal is out of dryer. Then stop drum.
7. Bring furnace temperature to 300° F. and hold at 300° F. to 400° F. for interval dryer is idle.
8. Bank fire so as to maintain a furnace temperature of 300° F. Fire so as not to stir up bed and cause sparks to carry to discharge end of dryer.

When stirring fire or slicing, cut speed of suction fan, keep discharge hood end of dryer always closed. When dryer is operating, allow as little outside air as possible to enter the dryer except over or through furnace and fire.

Keep inspection door on feed spout always closed except when inspecting feed and relieving choke in spout. Allow as little outside air as possible to pass into fan or short circuiting of air to fan inlet. Run dryer until free of all coal.

Clean out hood inlet to fan three or four times each day.

Clean out fan casing each day.

Inspect cyclone cone each day to see that there is no build-up on sides.

Keep gases at feed end of dryer below 1400° F. and at discharge end 850° F.

If wet coal runs out feed end of dryer, coal is too wet, too fine or too much feed.

1. Cut feed and speed up fan until relieved.

If coal is too wet with normal temperature, speed up fan. If still wet, raise heat to maximum temperature of 1400° F. If still too wet, cut feed.

If Coal Is Too Dry:

1. Cut speed of fan.
2. Inspect feed and see if normal.
3. Cut heat of furnace at feed end of dryer.

In Case of Quick Emergency Shutdown:

1. Cut feed.
2. By-pass gases to stack.
3. Stop suction fan.
4. Open doors to furnaces and cool off fire quickly.
5. Keep dryer revolving.

In Case of Fire in Hood Inlet to Suction Fan:

1. Stop suction fan.
2. Turn on water spray into hood.
3. Stop feed.
4. Stop dryer.
5. Open by-pass damper to stack.
6. Open furnace doors and cool furnace.
7. See that all doors and gates are closed at end of dryer. Keep as little air as possible entering discharge end of dryer.

In Case of Fire in Dryer:

1. Run fan at low speed to keep dryer free of gases.
2. Stop feed.
3. Keep dryer revolving.
4. Open by-pass damper.
5. Open furnace doors and cool furnace, and if necessary, pull fire.
6. See that all gates and doors at discharge end of dryer are closed. Keep as little air as possible entering dryer at any point.
7. Open water valves and flood dryer.

In Case of Power Failure:

1. By-pass gases to stack.
2. Open furnace doors and cool furnace, and if necessary, pull fire.

Caution

Do not weld or use cutting torch around dryer, conveyors, chutes or spouts feeding or discharging from dryer while dryer is operating

or fire in furnaces. Have the fire out of the furnace and be sure there is no dust in suspension.

Tests and analyses are under way to determine the amount of carbon dioxide and oxygen in the gases and what effect varying contents would have on retarding the fire hazard.

Power

The connected horsepower of all motors in the heat-drying plant is as follows:

Machine	Motor	Horsepower, Each Motor	Total Horsepower
Raw coal conveyor	1	60	60
Feeder conveyors	3	5	15
Dryers	3	60	180
Dryer fans	3	15	45
Dryer discharge conveyor	1	15	15
Dry coal conveyor	1	20	20
Stoker motors	3	1	3
Stoker fan motors	3	5	15

Total connected load, hp	353
Total connected load, kw	264
Average meter load, kw	216

Based on 135 tons per hour dry coal feed to three dryers, this gives 1.60 kw-hr. per ton of coal dried.

Plant Construction and Capacity

The dryer plant is housed in a structural steel frame, covered with galvanized corrugated iron, steel sash and steel doors, concrete floors and foundations.

Building is 60 by 93 ft. It is 20 ft. high over dryer and 50¼ ft. high over bins. The bin structure is 60 by 31 ft. and the dryer structure 60 by 62 ft.

The Pittsburgh Coal Co. has installed and is operating at the Champion No. 4, or Warden plant, two 90-in. by 40-ft. heat dryers, with capacity of 35 tons per hour; and at Champion No. 5, or Banning No. 1 plant, two 90-in. by 47-ft. dryers with capacity of 40 tons per hour. All dryers are of the L. R. Christie Co. BD type. The construction and operation are similar to the Champion No. 1 plant.

The seven dryers at the three plants have a potential drying capacity of 210 tons per hour.

DISCUSSION

(J. R. Campbell presiding)

J. R. Campbell, Pittsburgh, Pa.—We heard some able papers yesterday afternoon on the subject of the air cleaning of coal.[1] Some of us on the other side feel that wet washing is here to stay, and that the advancement in the art of coal cleaning will be along the lines of better drying of wet-washed coal, and it seems to me that a step in this direction is a step in the right direction. There is no trouble about the handling of the coarse sizes by natural drainage, and the fine sizes, if not low enough in moisture by mechanical drying or centrifugal drying, must be further dried by the use of heat dryers. Probably that will all straighten itself out in a few years when the coal consumer becomes used to wet-washed coal dried in the two ways described.

Mr. Parmley, could you translate some of those power and coal costs into dollars and cents per ton? Could you give us some idea of what this heat-drying cost is in terms of itself and in terms of feed coal?

S. M. Parmley.—We have no particular detailed cost of each item, but per ton of coal dried, it runs 7 c. per ton. That does not include depreciation and overhead. That is actual operating costs and supplies. Of course, it all depends upon the local conditions. Power, I have given as a unit and fuel in pounds of coal per ton of coal dried.

J. R. Campbell.—On this plant you are drying approximately what proportion of the feed coal?

S. M. Parmley.—About one-seventh.

J. R. Campbell.—That would add 1 c. to the cost of feed coal to the washer.

S. M. Parmley.—It is about 1 c. per ton on the feed coal to the washer.

B. M. Bird, Columbus, Ohio.—What range of sizes do you put in?

S. M. Parmley.—Just 0 by 3/8 in. Table 1 gives data relative to coal feed to three dryers at Champion No. 1. It is made up of 100 to 120 tons per hour of centrifugal dryer product and 20 to 25 tons of filtered product.

J. R. Campbell.—Filtered product is 48 mesh by 0 material?

S. M. Parmley.—The screen analysis gives feed to the dryer, 62 per cent. of 14 mesh and practically 8.5 per cent. −100 mesh.

F. A. Jordan, Youngstown, Ohio.—Am I correct in understanding that you would have as much trouble with coarse coal in the dryer as you would with the fine coal?

S. M. Parmley.—No, I do not think you would. We have not run any tests on anything over 3/8 in., but it is my opinion that if we had, say, −1-in. coal, we would not have the trouble with drying that we have now with the 0 to 3/8, as I think the coarser coal would have a tendency to scour the dryer flights and to give a larger capacity, and to give a better chance for air to pass through. When I referred to the coarse sizes, I meant the coarser sizes of 0 by 3/8-in., or the +14-mesh material. This is dealing principally with 0 by 3/8-in coal.

R. H. Schaller, Alquippa, Pa.—How long does the coal stay in the dryer, and does it pick up any oxygen?

S. M. Parmley.—The coal stays in the dryer about 20 min. from the time it enters the feed until it discharges. We have not run tests on oxygen as yet. Such tests are under way, but with regard to the volatile matter, we found that it did not change the volatile at all.

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[1] See pages 235 and 288.

B. M. Bird.—Have you tried putting any material from your flotation plant into the heat dryer?

S. M. Parmley.—Yes, what we are making at present all goes into the heat dryer.

B. M. Bird.—It has not given you any trouble?

S. M. Parmley.—Coal from the flotation plant has not given us any trouble at all, but, of course, at the present time, it is only about 20 per cent. of the total fine coal that is really going into the dryer; that is, total filter product.

B. M. Bird.—Oil is giving you no trouble whatever?

S. M. Parmley.—Oil is giving us no trouble.

J. R. Campbell.—The fact is, you are not using oil.

S. M. Parmley.—In experimental work at times we use some oil.

J. R. Campbell.—Will you translate into tons, for a 50-ton dryer, or for a 35-ton dryer, the amount of coal used in a day?

S. M. Parmley.—We use 15 lb. of coal fired per ton of dry coal. That is the way we interpret it.

E. G. Hill, Pittsburgh, Pa.—As you have pointed out, the amount of air passing through is as important as any other factor. Do you measure the air for control purposes?

S. M. Parmley.—We have no actual measurements, except working back from the water gage, the temperatures and the fan speeds. Air circulation runs between 15,000 and 17,000 cu. ft. per min. per dryer, depending on the moisture content of product and speed of dryer.

Dust Collection in Pneumatic Cleaning Plants

By Charles H. J. Patterson,* Pittsburg, Kansas

(Pittsburgh Meeting, September, 1930)

When coal is deposited on the decks of pneumatic tables, all fine particles clinging to the larger pieces are blown free by the air. Inasmuch as the air retains an appreciable residual velocity after passing through the bed of coal, it is capable of suspending particles of goodly size. This velocity is dissipated gradually and, unless confined, can readily extend beyond the limits of the plant. Today the ideal in the design and operation of pneumatic cleaning plants is not that they be no dustier than the ordinary tipple, but rather that they be as clean as an office or shop. This degree of perfection may not yet have been attained in any commercial plant, but it is not impossible of achievement.

Not until recently has the recirculation principle been applied to pneumatic separating plants. In the writer's estimation, it has not yet been proved entirely satisfactory in commercial installations. A serious obstacle lies in the fact that it is practically impossible to provide a truly closed circuit. The air circuits must be interrupted at certain points, if only for the introduction of raw material and for the recovery of the finished product. It is feasible to prevent the escape of polluted air at these points, but this can scarcely be accomplished without admitting additional air into the circuit. These additions of air are cumulative, which necessitates means for emitting to the atmosphere the excess while still carrying its objectionable burden, if provisions are not made for conditioning. If it must be provided at all, conditioning is frequently simpler and more economical if applied to the entire volume of air involved. This procedure eliminates the extra initial cost and operating expense of the closed system.

Crane Creek Experiments

The first noteworthy commercial installation of pneumatic equipment was made in 1922, at the Crane Creek plant of the American Coal Co. of Allegheny County, at McComas, W. Va. Six tables for cleaning coal between the limits of 1/8 and 3 in. comprised the initial installation. Dust was collected by hoods connected by a system of ducts to exhauster fans. In this installation the unit dust burden was small, owing to the removal of fines from the feed by vibrating screens.

* Engineer, Washing Division, Pittsburg Boiler & Machine Co.

After about one year of satisfactory operation, it was decided to extend the treatment to $-\frac{1}{8}$-in. coal and five additional tables of the Y-type were added. As these had relatively doubled the coal-cleaning capacity, with no proportionate increase in the quantity of air required, the dust burden of the air was increased many times as compared with the operation of the tables used on the larger coal.

Conditions were further aggravated by the use of aspirators to remove a large proportion of the fine dust from the coal before delivery to the tables. These were intended to give greater capacity to the tables in the cleaning of larger particles, on the assumption that the finest coal received no beneficial treatment on the decks. This resulted in increasing the dust escaping from the collectors to so great an extent that working conditions became unendurable and adjacent residences unfit for human habitation. Incidentally, aspirators have not been placed in any of the plants recently erected in the United States, although they are still being used in Canada, Australia and Great Britain.

When it became evident that the methods employed in the original dust-collecting installation could not cope with the altered conditions, a new piping system was installed by the B. F. Sturtevant Co. The investigations attendant on this last installation at McComas constitute in a large degree the basis of later study and development; yet today this plant is by no means modern in its method of dust collection.

Until recently, hoods for separators treating coal larger than $\frac{1}{8}$ in. were designed with open chimneys at the end farthest away from the feeding point of the table. This type is now obsolete in this country, the present tendency being toward totally enclosed equipment throughout the plant.

That the extremely fine "air float" particles in the smaller sizes cannot be reclaimed satisfactorily in centrifugal collectors is the most important lesson learned from the Crane Creek experience. The external discharge from the plant using such a system is highly objectionable though the interior conditions may be ideal. Filters employing cloth or other fine-mesh screening fabric, similar to those used in other industries, were considered at an early stage. These were intended for use where relatively small air volumes are employed; with large volumes the cost of these filters would have been prohibitive. Not only would they have complicated the plant but as direct filters they would have been subjected to quick ruination by the abrasive action of dust particles at high velocities.

With filters apparently demonstrated as not feasible, investigations were inaugurated in other directions. Scrubbing by jets and sprays of water and treatment with live steam were tried. Among other reasons for their failure, water processes produced a sludge, and this offset one of the frequently quoted advantages of dry cleaning over wet processes.

Birtley Process

Meanwhile, investigations conducted in England met with a greater measure of success. Centrifugal collectors were never seriously considered and complete removal of dust by filtration was held a primary requirement, owing to statutory restrictions on industrial discharges. A simple process, adequate and economical, was finally evolved by the Birtley Iron Co., Ltd. It was first installed in the Wardley plant of Messrs. John Bowes and Partners, Ltd., the first British plant to treat coal smaller than ⅛ in., and has been in operation about three years. Since then, similar equipment has been used in many plants in Europe, Australia and America. It is used by the American Coal Cleaning Corporation.

This system consists essentially of a precipitation chamber which receives the dust-laden air from the exhauster. Perforated plates form the floor of this chamber, and beneath each perforation is suspended a cylindrical filter tube formed of a fine-mesh fabric. These tubes connect with bins which otherwise are entirely enclosed except for valves for the withdrawal of dust.

In operation, the dust-impregnated air, entering the large expansion chamber at high velocity, quickly becomes almost motionless. The velocity of escape through the fabric is never more than 1/150 of the entering speed. With the velocity dissipated, all but the extremely fine particles fall through the tubes to the dust bins below and the removal of the air float residue constitutes the only actual filtering performed.

A film of dust finer than 100 mesh forms a protective lining on the inside of the tubes and assists in filtering, much as the *schmutzdecke* of the Imhoff sewage filter. As the thickness of this dust lining maintains a static balance, rapping devices and air reversal arrangements are unnecessary. The fabric used in the tubes is a standard English weave which permits no dust leakage and holds a back pressure in a static state indefinitely, once that pressure is established.

A new system combining filtration with centrifugal collection has been introduced recently by the Birtley company. Unfortunately, detailed information is not available at this time.

All-metal and Other Systems

Several all-metal systems, chiefly of the compounded centrifugal collector type, are now available. All these systems, which use ducts for collecting dust-laden air, practically follow the specifications determined upon in the Crane Creek installation.

There has been offered by the American Blower a new type of collector which is reported to have shown high efficiencies with various intensities of

coal dust. However, the initial cost is fairly high, and the resistance is high and appreciably increases the power consumption.

The Pangborn dust arrester, a cloth-screen type of filter, is being produced in comparatively large units, which are readily combinable in rows for multiple installations. Motor-driven screen shakers and solenoid-operated air-reversing mechanisms prevent ultimate blinding of the screens. In this system the over-all resistance is somewhat higher than the process last described. The largest installation of this equipment is at the Jerome (Pa.) plant of the Hillman Coal and Coke Co., which has a cleaning-plant capacity of 200 tons per hour and uses over 100,000 cu. ft. of filtered air per minute, it is said.

In the Peale-Davis system, dust is collected in a huge expansion compartment located above the plant. As the bottom of this chamber almost completely encloses the decks of the tables, all the air coming through the deck will enter the chamber. Hoppers are provided to receive the dust which precipitates as the velocity of the air is expanded in the large expansion zone. It is hardly conceivable that the extreme fines will precipitate in an expansion chamber of any reasonable size, but it is, of course, practicable to combine some type of filtering system where complete recovery is essential. The air volumes required for treating 4 by 0 in. coal are so great in proportion to the amount of the finest dust that the unit dust burden is not nearly so great as in the processes treating ⅛-in. coal separately.

For volumes of 25,000 cu. ft. per min. or less, the larger sizes of planing-mill type of exhausters are generally used. These are strong in construction and do not consume power excessively; but they are not made with wheels larger in diameter than 7 or 8 ft. In some instances, it is advantageous to split the system into two or more independent circuits, but this inevitably results in increased initial cost and operating expense. Because of these facts, the so-called multivane types of fans are coming to be regarded as standard practice for large air volumes. Though these are not of such strong construction as those of the planing-mill type, the units in use for several years show no serious wear and have effected economies in power consumption.

Determination of Shapes of Particles and Their Influence on Treatment of Coal on Tables*

By H. F. Yancey,† Seattle, Wash.

(Pittsburgh Meeting, September, 1930)

Poor results in coal washing and in ore concentration are sometimes attributed to the shape of the particles in the feed. It is well known that the shape of a particle influences its rate of fall in water. This is evidenced by the value of the constant C in the Rittinger equation for the velocity V of settling in water, $V = C\sqrt{D(d - L)}$, where C has the following values: 2.73 for roundish grains, 2.37 for long grains and 1.92 for flat grains.

This report, which is based upon data obtained during the course of coal-washing investigations conducted by the U. S. Bureau of Mines in cooperation with the College of Mines of the University of Washington, gives the results of a method or procedure of screening in which the particles are separated into three shapes and shows the deportment of these shapes when coal is treated on a coal-washing table. The method consisted in first screening the coal with Tyler standard square-hole sieves, followed by screening tests of each of these products with screens of rectangular opening. For this purpose Tyler "Ton-Cap" screens were used; each square-hole product was subsequently sized with two Ton-Cap sieves so selected that their shortest dimensions were approximately 75 and 50 per cent., respectively, of the opening of the square-mesh sieves. For example: A product which is through 8 mesh (0.093 in.) and on 10 mesh (0.065 in.) is screened on slotted sieves which have openings 75 and 50 per cent. of the 0.065-in. opening in one direction and an opening of not less than 0.065 in. in the other direction. For convenience in discussion, the particles retained on the first slotted screen are termed "cubical"; those retained on the second slotted screen are termed "prismatic" and those passing the second screen are called "flat." (See Fig. 1.) The square-mesh screens separate particles according to two dimensions, length and breadth; the slotted screens divide them further according to the third dimension, or thickness. Prismatic particles have a mass of about two-thirds, and flat particles of about one-half, that of cubical particles of the same square-mesh size. The screen-

* Published by permission of the Director, U. S. Bureau of Mines.

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† Acting Supervising Engineer, U. S. Bureau of Mines, Northwest Experiment Station.

ing procedure described is the same in principle as has been used for determining the flakiness of ores at the Mississippi Valley Experiment Station of the Bureau of Mines.[1] Table 1 shows the screen scale and the dimensions of the slotted screens used.

TABLE 1.—*Screen Scale for Determining Shape of Particles*

Square Opening		Rectangular Opening		
Sized Product, Inches and Mesh	Opening of Last Size, Inches	Specified Width, Opening, Inches	Nearest Width, Opening, Inches	Stock Ton-Cap No.
0.525 to 0.371	0.371	0.278	0.280	389
		0.185	0.181	407
0.371 to 3	0.263	0.197	0.194	607
		0.131	0.128	1196
3 to 4	0.185	0.139	0.142	767
		0.093	0.091	599
4 to 6	0.131	0.098	0.095	446
		0.066	0.064	740
6 to 8	0.093	0.069	0.071	556
		0.046	0.048	497
8 to 10	0.065	0.0487	0.048	497
		0.0325	0.0317	819
10 to 14	0.046	0.0345	0.0347	817
		0.0230	0.0225	422
14 to 20	0.0328	0.0246	0.0255	786
		0.0164	0.0157	143
20 to 28	0.0232	0.0174	0.0177	494
		0.0116	0.0116	176
28 to 35	0.0164	0.0123	0.0123	155
		0.0082	0.0087	190

SCREENING PROCEDURE

A sample of coal from Colorado, which had been crushed to pass a ¾-in. square-hole screen, was investigated by the method outlined. Four solutions having specific gravities of 1.30, 1.38, 1.50 and 1.70, were used to separate the original sample into five products. Each of these products was then sized with the square-hole screens and the slotted screens as described above. The results of this separation are shown in Table 2. In each specific gravity fraction, coarser sizes contain a greater proportion of cubical particles than the finer sizes. The proportion of flat particles in each size increases with the specific gravity of the components of the raw coal. The weighted average percentage of cubical, prismatic and flat particles for each specific-gravity fraction is shown at the bottom of Table 2. The cubical particles in the fraction having a specific gravity less than 1.30 amount to 67.4 as compared with 61.1,

[1] W. H. Coghill, O. W. Holmes and A. B. Campbell: Determination of Flakiness of Ores. U. S. Bur. Mines *Repts. of Investigations* Ser. 2899 (1928).

54.1, 53.3 and 35.2 per cent., respectively, in the fractions of higher specific gravities. The tendency toward flat shapes or flakiness increases as the specific gravity of the components increases. The component with a specific gravity under 1.30 contains nearly twice as much cubical material as the component over 1.70 specific gravity but only one-fifth as much flat material. These data give numerical expression to the common observation that bone and refuse contain more flat shapes than does coal, and that this particular coal tends to assume cubical forms.

Influence of Size and Shape in Tabling

The method of screening outlined was used as a measuring stick to determine whether or not shape of particle is a factor in cleaning coal

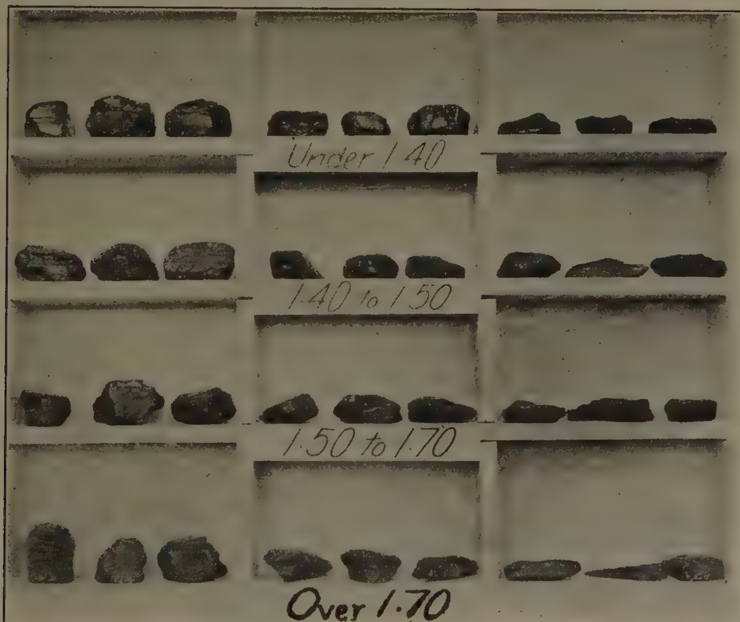

FIG. 1.—CUBICAL, PRISMATIC AND FLAT PARTICLES IN FOUR SPECIFIC GRAVITY FRACTIONS OF A RAW COAL.

on the wet coal-washing table. It is well known that specific gravity and size are important factors in all methods of concentration including table concentration.

In tabling unsized coal through a given limiting screen, size interferes to some extent with the separation according to specific gravity.[2] A por-

[2] B. M. Bird: Sizing Action of Coal-washing Table. U. S. Bur. Mines *Repts. of Investigations* Ser. 2755 (1926).

A. W. Fahrenwald: Hydraulic Classification. U. S. Bur. Mines *Tech. Paper* 403 (1927) 3.

R. H. Richards and C. E. Locke: Textbook of Ore Dressing, Ed. 2, 213. New York, 1925. McGraw-Hill Book Co.

TABLE 2.—*Amount of Cubical, Prismatic and Flat Particles in Specific-gravity Products of Screen Sizes of Raw Coal*

Opening, Inches and Mesh	Shape	Under 1.30	1.30 to 1.38	1.38 to 1.50	1.50 to 1.70	Over 1.70	All Specific Gravities Combined
		Weight, Per Cent.	Weight, Per Cent.	Weight, Per Cent.	Weight, Per Cent.	Weight, Per Cent.	Weight, Per Cent.
On 0.525		0.4	1.0	1.3	1.1	1.6	0.8
0.525 to 0.371	Cubical	70.1	69.9	64.0	63.2	46.2	66.6
	Prismatic	28.3	28.4	33.8	34.0	39.6	30.6
	Flat	1.6	1.7	2.2	2.8	14.2	2.8
0.371 to 3	Cubical	82.1	77.4	77.0	72.1	57.3	77.4
	Prismatic	16.1	20.6	20.2	25.0	32.4	20.0
	Flat	1.8	2.0	2.8	2.9	10.3	2.6
3 to 4	Cubical	66.7	60.9	51.4	54.6	34.8	59.4
	Prismatic	31.6	36.6	44.0	40.3	47.9	36.9
	Flat	1.7	2.5	4.6	5.1	17.3	3.7
4 to 6	Cubical	74.9	62.7	50.6	55.5	35.1	65.0
	Prismatic	21.1	31.7	38.1	31.8	37.3	27.8
	Flat	4.0	5.6	11.3	12.7	27.6	7.2
6 to 8	Cubical	67.4	46.2	44.3	34.8	20.7	52.6
	Prismatic	28.0	44.9	43.6	50.0	43.3	37.4
	Flat	4.6	8.9	12.1	15.2	36.0	10.0
8 to 10	Cubical	78.3	67.7	60.1	51.4	33.9	69.0
	Prismatic	16.7	27.1	30.9	35.5	39.7	23.5
	Flat	5.0	5.2	9.0	13.1	26.4	7.5
10 to 14	Cubical	61.0	53.6	39.8	39.7	27.5	53.3
	Prismatic	33.4	37.7	43.9	41.0	50.0	37.2
	Flat	5.6	8.7	16.3	19.3	22.5	9.5
14 to 20	Cubical	41.2	36.1	25.5	24.4	21.4	35.6
	Prismatic	51.7	55.7	64.9	60.2	58.6	55.2
	Flat	7.1	8.2	9.6	15.4	20.0	9.2
20 to 28	Cubical	29.8	25.2	19.5	15.8	32.2	27.4
	Prismatic	47.6	44.4	45.3	49.3	28.6	43.8
	Flat	22.6	30.4	35.2	34.9	39.2	28.8
Through 28		12.8	11.9	13.8	13.3	18.8	13.2
Weighted average, all sizes	Cubical	67.4	61.1	54.1	53.3	35.2	60.5
	Prismatic	28.0	33.5	37.8	36.9	41.8	32.6
	Flat	4.6	5.4	8.1	9.8	23.0	6.9
All sizes		42.4	29.6	13.9	6.4	7.7	100.0

tion of coarse particles of bone and refuse is discharged with the washed coal merely because the particles are coarse. This condition, of course, may be reversed and the effect of size can be made to aid instead of hinder the separation by hydraulic classification of the feed prior to tabling.[3] In a classified feed the particles decrease in size with increase in specific gravity.

Although shape is often listed along with size and specific gravity as one of the determining factors in concentration, its true significance and importance are not well understood because it is difficult to evaluate.

[3] B. M. Bird and H. F. Yancey: Hindered-settling Classification of Feed to Coal-washing Tables. *Trans.* A. I. M. E., Coal Div. (1930) 250.

Recent experiments by the Bureau of Mines with spherical and angular grains of quartz and other minerals have shown that shape is a factor of some importance in table concentration.[4] Since the earlier work, referred to above, at the Northwest Experiment Station had evaluated the influence of specific gravity and size in tabling coal, it appeared desirable to investigate the effect of shape by a table concentration test of coal.

Test Procedure

The coal treated in the tabling test was obtained by passing a 40-ton lot of the ¾-in. coal, of Table 2, over a 407 Ton-Cap (approximately 3 mesh) screen; the undersize, with all the fine sizes present, was used as the feed. Table 3 shows the specific gravity analysis of the feed and the final results of the washing test. The test was made at the rate of 4.9 tons of feed per hour and the table adjustments were as follows: stroke, 1 in., number of strokes a minute, 268; cross slope in inches per foot, 1.08; elevation of refuse end above head-motion end in inches, 3.22; ratio of weight of water to weight of coal, 2.30. After the table had been adjusted to give the best visual operation, a time sample of the products discharged from the table was collected in 1-ft. zones, using the special sampling launder shown in Fig. 2. The products in each two successive zones, with one exception, were combined for subsequent examination, as shown in Table 3.

Table 3.—*Tabling Test of Unsized Feed*

Operation	Specific Gravity and Zones[a]	Weight, Per Cent.	Ash,[b] Per Cent.	Cumulative Weight, Per Cent.	Cumulative Ash,[b] Per Cent.
Specific-gravity analysis of feed..........	Under 1.38	72.8	7.0	72.8	7.0
	1.38 to 1.50	13.3	20.2	86.1	9.0
	1.50 to 1.70	6.5	34.5	92.6	10.8
	Over 1.70	7.4	67.1	100.0	15.0
Table products.......	1 and 2	2.8	8.5	2.8	8.5
	3 and 4	12.3	8.0	15.1	8.1
	5 and 6	18.3	8.9	33.4	8.5
	7 and 8	21.1	9.3	54.5	8.8
	9 and 10	18.0	9.3	72.5	8.9
	11 and 12	10.6	10.4	83.1	9.1
	13, 14 and 15	3.5	18.1	86.6	9.5
	16 and 17	1.6	24.9	88.2	9.8
	18 and 19	3.2	34.2	91.4	10.6
	20 and 21	8.6	61.3	100.0	15.0

[a] One-foot zones.
[b] Moisture-free basis.

[4] A. W. Fahrenwald and W. F. Meckel: The Relation of Table Feed Preparation to Table Efficiency. U.S. Bur. Mines *Repts. of Investigations* Ser. 2949 (1929).

Float and sink tests were made on each of the zonal products, using baths of 1.38, 1.50 and 1.70 specific gravities, resulting in four products designated as "coal," "light bone," "heavy bone" and "refuse." The particles in each specific gravity component of each zonal product were then separated according to size and shape by screening.

The screen-sizing tests of the specific-gravity fractions of the zonal products using the square-hole sieves provide information about the influence of size. The screening tests with slotted sieves supply informa-

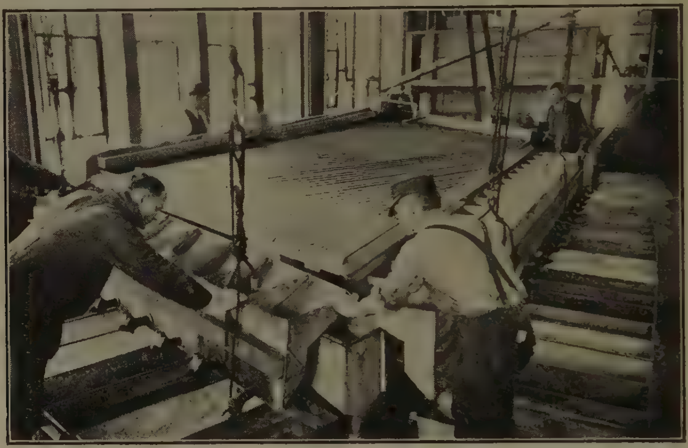

Fig. 2.—Launder for taking zonal samples of table products.

tion about shape. The data (Table 4), showing the sizing effect of tabling, will be presented first. The average diameters of the coal, light bone, heavy bone and refuse particles were approximated by screen-sizing and weighing the specific-gravity fractions found in each zonal product. Sieves in the fixed ratio of the square root of two were used.

Influence of Size

Table 4 gives data from one of seven tabling tests made on this coal. Despite the fact that this was the best test, the adverse effect of size on the separation is marked. With the exception of the first few zones where the rush of water across the deck prevents proper stratification, it will be observed that, in general, the coarsest material of any specific-gravity fraction is discharged first. The average diameter of particles decreases with distance from zone 1. In a given zone, the average diameters increase with increase in specific gravity, except, however, for the refuse, which is smaller. This is due partly to its disintegration in the float and sink testing and partly to the discharge of a high proportion of fine refuse (finer than 48 mesh) in the first 17 zones. This effect is especially pronounced in zones 1 to 4, inclusive.

TABLE 4.—*Size of Specific Gravity Products in Zonal Cuts from Table with Unsized Feed*

Product	Coal, Under 1.38 Sp. Gr.		Light Bone, 1.38 to 1.50 Sp. Gr.		Heavy Bone, 1.50 to 1.70 Sp. Gr.		Refuse, Over 1.70 Sp. Gr.	
Feed,[a] per cent.	72.9		12.9		6.8		7.4	
Zone No.	Weight, Per Cent.	Average Diameter, Mm.	Weight, Per Cent.	Average Diameter, Mm.	Weight, Per Cent.	Average Diameter, Mm.	Weight, Per Cent.	Average Diameter, Mm.
1 and 2	3.4	1.8	1.3	0.4	1.0	0.4	0.7	0.1
3 and 4	15.5	4.6	6.4	5.0	1.1	4.2	0.2	0.4
5 and 6	22.1	3.8	15.3	4.6	4.7	4.5	0.4	2.4
7 and 8	24.7	2.9	21.2	3.6	7.1	3.9	0.5	2.4
9 and 10	20.5	1.9	19.0	2.7	8.2	2.9	0.7	2.0
11 and 12	11.0	1.0	15.8	1.6	6.7	1.8	0.8	1.1
13 to 15, incl.	2.1	0.7	9.9	1.7	9.4	2.0	1.0	1.2
16 and 17	0.5	0.6	4.6	1.3	8.9	1.8	1.3	1.3
18 and 19	0.1	1.7	4.6	2.8	31.3	3.5	4.6	3.0
20 and 21	0.1	1.9	1.9	0.9	21.6	2.2	89.8	2.5
Weighted average.....	100.0	2.8	100.0	2.9	100.0	2.8	100.0	2.5

[a] Calculated from float and sink of zonal products after screen sizing.

About one-eighth, or 12.9 per cent., of the total quantity of heavy bone in the feed is discharged in 6 zones (3 to 8 inclusive) near the head-motion end, and this is the coarsest of the heavy bone in the feed. This discharge of coarse, heavy bone with the coal decreases the efficiency of the separation at low ash content. However, not all of the coarse, heavy bone is discharged in the first zones of the table. The average diameter of the heavy bone discharged on the washed coal side of the table (zones 1 to 17 inclusive) is 2.6 mm. and amounts to 47.1 per cent. of the quantity in the feed; but that discharged on the refuse end (zones 18 to 21 inclusive) has an average diameter of nearly 3.0 mm. and represents 52.9 per cent. of that present in the feed. The reason for this condition was found by determining the ash content of the heavy bone in each zonal product, which, as would be expected, increases in each successive zone. The average ash content of the heavy bone in all the zones on the side of the table was 31 per cent.; from the end it was 37 per cent. Obviously, that portion of the heavy bone which is lowest in specific gravity is discharged first. This same general tendency is exhibited by the other three specific-gravity fractions.

INFLUENCE OF SHAPE

Tables 5, 6, 7 and 8 show the influence of shape on table concentration as determined by the screening tests with slotted screens. A separate table is presented for each specific-gravity fraction. Only the square-mesh products between 3 and 35 mesh were screened with slotted sieves,

and because the original feed was through a slotted or rectangular opening, the size larger than 3 mesh was not separated as to shape. This size amounted to only 1.1 per cent. of the total feed. The portion of the feed finer than 35 mesh amounted to 6.9 per cent.; hence, of the total feed, 92.0 per cent. was screened with slotted sieves. The figures shown in the tables represent the percentage of each shape delivered in each zonal cut for each of the square-mesh sizes. Each individual shape of each square-mesh size is considered as 100 per cent. The weighted average of all sizes from 3 to 35 mesh is shown at the bottom of each table.

TABLE 5.—*Distribution of Cubes, Prisms and Flakes of Coal Under* 1.38 *Specific Gravity and of Different Sizes, in Zonal Cuts of Table**

Zones		1 and 2	3 and 4	5 and 6	7 and 8	9 and 10	11 and 12	13 to 15	16 and 17	18 and 19	20 and 21	All Zones
3 to 4 mesh	Cubical	3.3	49.1	28.7	15.9	2.4	0.2	0.1		0.2	0.1	7,704
	Prismatic	2.6	36.9	31.7	25.3	3.1	0.2	0.1		0.1		8,808
	Flat	2.4	22.8	34.1	35.2	5.1	0.4					452
4 to 6 mesh	Cubical	2.1	28.3	38.4	22.4	8.2	0.4	0.1		0.1		16,846
	Prismatic	1.2	17.9	34.8	33.0	12.2	0.8	0.1				4,997
	Flat	1.1	16.5	28.8	36.3	16.5	0.8					707
6 to 8 mesh	Cubical	1.0	14.7	36.4	29.5	17.0	1.3	0.1				9,728
	Prismatic	0.6	9.8	36.0	35.5	16.0	2.1					4,492
	Flat	1.0	8.3	29.4	39.5	19.2	2.6					650
8 to 10 mesh	Cubical	0.6	5.0	22.1	34.4	32.1	5.6	0.2				9,479
	Prismatic	0.8	4.9	25.9	31.6	29.0	7.6	0.2				2,571
	Flat	3.3	6.8	33.3	23.8	25.6	7.0	0.2				457
10 to 14 mesh	Cubical	0.3	0.8	4.3	39.8	39.5	14.8	0.5				9,240
	Prismatic	0.9	0.7	4.8	29.2	43.0	20.1	1.2	0.1			5,611
	Flat	1.0	0.6	4.4	21.9	47.9	22.7	1.4	0.1			1,143
14 to 20 mesh	Cubical	1.9	0.9	4.1	17.5	36.7	33.7	4.2	0.7	0.2	0.1	3,393
	Prismatic	2.8	0.7	3.0	20.2	36.4	29.7	6.1	0.9	0.2		3,474
	Flat	4.9	1.9	2.6	17.4	34.2	31.6	6.7	0.7			430
20 to 28 mesh	Cubical	3.5	1.0	2.7	7.2	32.9	36.4	12.6	2.4	1.0	0.3	1,819
	Prismatic	5.2	0.8	2.1	7.3	29.8	39.2	12.4	2.4	0.5	0.3	2,537
	Flat	6.1	2.2	3.7	10.8	25.8	30.9	17.7	2.3	0.3	0.2	1,014
28 to 35 mesh	Cubical	9.1	1.0	2.4	10.4	19.6	38.3	12.9	4.7	1.6		1,149
	Prismatic	11.2	1.2	2.8	7.7	23.8	37.6	10.9	3.8	1.0		2,353
	Flat	16.0	1.5	3.6	6.5	20.3	40.5	9.0	2.2	0.4		975
3 to 35 mesh	Cubical	1.7	17.8	25.2	26.4	20.2	7.4	1.0	0.2	0.1		59,358
	Prismatic	2.5	13.8	20.9	25.8	21.2	12.7	2.5	0.5	0.1		34,843
	Flat	5.0	6.1	14.3	21.9	26.4	19.9	5.5	0.8	0.1		5,828

* Figures given represent percentage of each shape delivered in zonal cuts; in each horizontal line the figures total 100 per cent. Under "all zones" is shown the total weight in grams of each shape for each size.

Attention is now directed to Table 5, which shows the distribution made by the table of cubical, prismatic and flat particles of coal in the different zones. Considering the 3 to 4-mesh size of coal, it will be observed that 49.1 per cent. of all the cubical shapes is discharged in zones 3 and 4. These same zones show a smaller percentage of prismatic and flat shapes; the figures are 36.9 and 22.8, respectively. Evidently, then, the more flaky particles, prismatic and flat, are carried out farther on the

deck than are the cubical particles of the same square-mesh size. This supposition is confirmed by observing that, though the maximum percentage of cubical particles of the 3 to 4-mesh size is discharged in zones 3 and 4, the maximum percentage of flat particles of this same size is found in 7 and 8, 4 ft. farther along the table deck. A similar condition will be noted for the 4 to 6-mesh size of coal, but because this product is of smaller size, a maximum proportion of cubes is discharged in zones 5

TABLE 6.—*Distribution of Cubes, Prisms and Flakes of Light Bone of* 1.38 *to* 1.50 *Specific Gravity and of Different Sizes in Zonal Cuts of Table*

Zones		1 and 2	3 and 4	5 and 6	7 and 8	9 and 10	11 and 12	13 to 15	16 and 17	18 and 19	20 and 21	All Zones
3 to 4 mesh	Cubical	0.2	22.5	38.7	23.8	6.6	0.7	0.5	0.1	6.4	0.5	1354
	Prismatic	0.1	17.9	34.4	26.7	11.2	1.0	1.6	0.7	6.3	0.1	1667
	Flat		12.9	34.5	31.0	14.7	1.7	2.6	0.9	1.7		116
4 to 6 mesh	Cubical		10.7	28.3	32.2	16.1	2.9	3.7	1.3	4.7	0.1	2846
	Prismatic		6.0	21.4	33.3	23.4	5.7	6.9	1.6	1.6	0.1	1033
	Flat		6.5	20.9	34.3	26.4	7.0	4.5	0.4	0		201
6 to 8 mesh	Cubical		3.4	16.0	27.1	29.8	10.3	8.7	2.4	2.3		1489
	Prismatic		1.9	12.6	29.5	27.0	18.2	8.6	1.7	0.5		907
	Flat		1.5	12.7	30.4	24.9	24.9	4.6	0.5	0.5		197
8 to 10 mesh	Cubical		0.7	4.6	28.1	34.9	17.7	9.6	2.8	1.5	0.1	1622
	Prismatic		0.3	3.6	26.7	28.9	25.3	11.3	3.2	0.6	0.1	861
	Flat		0.3	2.4	29.2	30.6	28.9	6.8	1.5	0.3		294
10 to 14 mesh	Cubical			1.6	9.0	25.1	35.5	16.9	7.0	4.0	0.9	633
	Prismatic			0.8	10.5	27.3	34.3	16.5	6.7	3.1	0.8	750
	Flat			0.8	12.9	21.7	41.7	14.6	6.2	1.3	0.8	240
14 to 20 mesh	Cubical			0.6	4.7	16.1	31.2	21.0	12.0	9.5	4.9	509
	Prismatic			0.3	5.0	15.9	34.5	21.4	11.7	6.9	4.3	598
	Flat			1.1	6.6	20.9	39.5	19.8	7.7	2.2	2.2	91
20 to 28 mesh	Cubical	0.3		0.3	3.0	10.2	25.8	20.7	14.9	13.3	11.5	295
	Prismatic	0.6		0.4	3.0	8.9	32.9	22.7	12.1	9.4	10.0	471
	Flat	1.2		1.2	6.1	11.8	27.8	29.9	11.0	4.1	6.9	245
28 to 35 mesh	Cubical						22.7	21.6	19.3	20.5	15.9	176
	Prismatic						18.0	21.1	25.1	18.3	17.5	350
	Flat						33.9	21.4	21.4	13.5	9.8	112
3 to 35 mesh	Cubical	0.4	7.5	18.6	24.4	20.4	11.5	7.9	3.4	4.7	1.2	8924
	Prismatic	0.1	5.7	14.3	21.3	19.1	17.1	10.6	5.1	4.6	2.1	6637
	Flat	0.2	2.1	8.0	20.2	20.7	26.3	12.8	5.3	2.3	2.1	1496

* Figures given represent percentage of each shape delivered in zonal cuts; in each horizontal line the figures total 100 per cent. Under "all zones" is shown the total weight in grams of each shape for each size.

and 6 instead of 3 and 4. A further inspection of Table 5 shows that the coal finer than 10 mesh discharged in the first six zones of the table is flaky in character; the percentage of prismatic and flat shapes discharged exceeds the percentage of cubical shapes in these fine sizes. This tendency of the fine sizes is not enough to change the general picture, as shown by all sizes 3 to 35 mesh, at the bottom of the table, because they are present in much smaller amount than are the coarser sizes. With zones 9 and 10, considering all sizes 3 to 35 mesh, the percentage of flat particles discharged begins to exceed the percentage of cubical particles.

This condition persists for the remaining zones of the table. The general tendency for cubes of coal to be discharged in greater proportion than the more flaky, prismatic and flat shapes is clearly brought out by cumulating the respective percentages at any given zone. For example, up to zones 7 and 8 the proportion of cubical, prismatic and flat particles discharged in the 3 to 35-mesh material is 71.1, 63.0, and 47.3 per cent., respectively.

TABLE 7.—*Distribution of Cubes, Prisms and Flakes of Heavy Bone of* 1.50 *to* 1.70 *Specific Gravity and of Different Sizes in Zonal Cuts of Table*

Zones		1 and 2	3 and 4	5 and 6	7 and 8	9 and 10	11 and 12	13 to 15	16 and 17	18 and 19	20 and 21	All Zones
3 to 4 mesh	Cubical		4.3	15.4	11.8	5.1	1.7		0.2	31.0	30.5	533
	Prismatic		2.5	9.9	11.6	5.9	1.6	1.6	1.3	50.7	14.9	926
	Flat		2.1	12.8	11.7	8.5	2.1	4.3	3.2	50.0	5.3	94
4 to 6 mesh	Cubical		1.7	9.6	13.0	8.8	2.3	3.4	2.6	49.8	8.8	1266
	Prismatic		1.1	6.6	12.0	10.1	3.6	8.8	7.4	45.4	5.0	557
	Flat		1.5	6.8	15.0	9.0	5.3	15.0	10.5	34.6	2.3	133
6 to 8 mesh	Cubical		0.3	3.9	9.8	11.1	5.0	9.9	8.4	44.4	7.2	694
	Prismatic		0.2	2.2	8.9	11.7	6.2	16.3	13.9	34.7	5.9	504
	Flat		0.7	2.9	13.6	15.1	13.7	18.0	14.4	18.0	3.6	139
8 to 10 mesh	Cubical		0.0	1.2	5.6	11.8	7.7	13.4	13.0	33.4	13.9	662
	Prismatic		0.4	0.8	4.7	11.4	11.5	15.2	14.5	25.9	15.6	490
	Flat		0.0	1.2	7.5	16.2	19.4	16.2	13.1	15.7	10.7	160
10 to 14 mesh	Cubical			0.6	3.2	9.4	8.9	12.4	13.6	26.2	25.7	339
	Prismatic			0.2	2.4	9.5	10.2	13.3	12.5	21.3	30.6	422
	Flat			0.7	2.2	11.6	15.9	13.0	14.5	13.8	28.3	138
14 to 20 mesh	Cubical				1.3	7.1	8.4	10.7	14.0	20.5	38.0	308
	Prismatic				1.2	6.0	8.0	11.6	12.8	16.8	43.6	413
	Flat				1.6	9.5	11.1	12.7	11.1	15.9	38.1	63
20 to 28 mesh	Cubical				0.5	3.8	6.2	11.1	15.4	15.4	47.6	208
	Prismatic				0.6	4.1	6.5	12.5	12.8	13.6	49.9	337
	Flat				1.5	5.7	8.8	26.4	9.8	12.5	35.3	193
28 to 35 mesh	Cubical								12.5	20.8	66.7	72
	Prismatic								10.4	15.6	74.0	154
	Flat								13.1	19.7	67.2	61
3 to 35 mesh	Cubical		1.1	5.9	8.5	8.7	4.7	7.3	7.5	37.4	18.9	4082
	Prismatic		0.8	3.8	6.8	8.0	5.8	9.6	9.5	32.9	22.8	3803
	Flat		0.5	2.9	7.0	10.2	10.7	15.5	11.4	21.2	20.6	981

* Figures given represent percentage of each shape delivered in zonal cuts; in each horizontal line the figures total 100 per cent. Under "all zones" is shown the total weight in grams of each shape for each size.

The light bone of 1.38 to 1.50 specific gravity, shown in Table 6, exhibits the same general tendency as does the coal under 1.38 specific gravity, but the tendency for flaky material of a given square-mesh size to be carried out farther along the deck is much less pronounced. Nevertheless, the combined results for all sizes show that a greater percentage of cubes than prismatic and flat shapes is discharged in the first zones of the table.

Table 7 shows the deportment of the different shapes of heavy bone. Here the general tendency in the first eight zones is for the cubical material to be discharged in a greater proportion than the prismatic

and flat material. However, the differences in the percentages are less striking than in the case of the coal and light bone, and beginning with zones 9 and 10 the flaky particles come off in greater proportion than do the cubical particles. The general effect of shape is more erratic; in fact, all sizes smaller than 4 mesh beginning with zones 7 and 8 show a greater proportion of flat material discharged than of cubical material, with the

TABLE 8.—*Distribution of Cubes, Prisms and Flakes of Refuse Over* 1.70 *Specific Gravity and of Different Sizes in Zonal Cuts of Table*

Zones		1 and 2	3 and 4	5 and 6	7 and 8	9 and 10	11 and 12	13 to 15	16 and 17	18 and 19	20 and 21	All Zones
3 to 4 mesh	Cubical			0.6	0.3	0.6				3.1	95.4	325
	Prismatic			0.8	0.4	0.3				3.4	95.1	741
	Flat			0.5	0.0	0.0				6.2	93.3	194
4 to 6 mesh	Cubical			0.5	0.9	0.7	0.1	0.2	0.3	7.3	90.0	914
	Prismatic			0.3	0.6	0.4	0.3	0.3	0.6	9.0	88.5	719
	Flat			0.0	0.3	0.3	0.0	0.3	0.8	8.2	90.1	364
6 to 8 mesh	Cubical			0.5	1.5	1.8	0.8	1.3	1.8	12.3	80.0	390
	Prismatic			0.2	0.5	0.9	0.5	1.0	1.4	8.4	87.1	581
	Flat			0.0	0.3	0.7	0.3	1.4	1.0	7.1	89.2	296
8 to 10 mesh	Cubical				1.0	1.9	1.5	1.2	1.5	7.6	85.3	523
	Prismatic				0.5	1.2	1.4	1.2	1.9	6.0	87.8	418
	Flat				0.5	1.0	1.0	1.4	1.9	5.8	88.4	208
10 to 14 mesh	Cubical					0.9	0.9	0.9	0.9	2.6	93.8	456
	Prismatic					0.4	0.6	0.5	0.5	1.4	96.6	795
	Flat					0.3	0.5	0.5	0.5	1.1	97.1	369
14 to 20 mesh	Cubical					1.3	2.6	1.3	1.3	3.2	90.3	154
	Prismatic					0.4	0.6	0.6	0.6	0.8	97.0	494
	Flat					0.0	0.9	0.0	0.9	0.9	97.3	112
20 to 28 mesh	Cubical						1.6	0.8	1.6	2.5	93.5	123
	Prismatic						0.7	1.0	0.7	1.0	96.6	305
	Flat						0.3	1.5	0.6	0.6	97.0	334
28 to 35 mesh	Cubical										100.0	78
	Prismatic										100.0	243
	Flat										100.0	132
3 to 35 mesh	Cubical			0.3	0.7	1.0	0.7	0.7	0.9	6.2	89.5	2963
	Prismatic			0.2	0.3	0.5	0.5	0.5	0.7	4.2	93.1	4296
	Flat			0.1	0.2	0.3	0.3	0.7	0.7	4.1	93.6	2009

* Figures given represent percentage of each shape delivered in zonal cuts; in each horizontal line the figures total 100 per cent. Under "all zones" is shown the total weight in grams of each shape for each size.

exception of zones 18 and 19. This trend is probably due to the fact that a heavy bone contains a higher percentage of flat particles than either of the two lighter specific-gravity fractions. It has been pointed out previously in Table 2 that the proportion of cubes decreases and the proportion of flakes increases with increase in specific gravity. Zones 18 and 19 contain a greater percentage of the cubical particles discharged than of the other two more flaky shapes in all except one size. This material has a large average diameter, as previously shown in Table 4.

Table 8 shows the results for refuse material whose specific gravity is more than 1.70. Quite naturally, only small amounts of this fraction

are discharged in the first zones of the table. Nevertheless, here again the proportion of cubical material discharged exceeds the proportion of flaky shapes. This condition exists up to and including zones 11 and 12, and again occurs in zones 16 to 19, inclusive; in fact, the cumulative proportion of cubical particles discharged up to and including zones 18 and 19, in every size except the 3 to 4 mesh, is greater than is the proportion of flat shapes.

TABLE 9.—*Ash Contents of Cubical, Prismatic and Flat Particles in Specific Gravity Products*

Product	Size, Mesh	Cubical, Ash, Per Cent.[a]	Prismatic, Ash, Per Cent.[a]	Flat, Ash, Per Cent.[a]
Zones 3 and 4,[b] under 1.38 sp. gr.	On 3	8.9	8.5	8.4
	3 to 4	7.9	7.8	8.0
	4 to 6	7.2	7.3	7.3
	6 to 8	6.3	6.1	6.6
	8 to 10	5.4	5.2	5.5
	10 to 14	6.9	6.9	7.0
Zones 13 to 15, under 1.38 sp. gr.	10 to 14	13.1	12.5	12.4
	14 to 20	10.9	10.3	10.4
	20 to 28	9.1	8.8	8.4
Zones 13 to 15, 1.38 to 1.50 sp. gr.	3 to 4	21.6	23.2	24.8
	4 to 6	23.7	24.7	25.8
	6 to 8	24.5	25.7	26.5
	8 to 10	24.1	24.6	25.7
	10 to 14	22.7	22.8	23.8
	14 to 20	21.2	21.4	22.9
	20 to 28	19.6	20.1	20.1
Zones 13 to 15, 1.50 to 1.70 sp. gr.	4 to 6	33.2	33.2	35.2
	6 to 8	31.6	33.2	36.0
	8 to 10	31.6	33.1	35.4
	10 to 14	30.2	30.5	32.9
	14 to 20	26.8	27.4	30.3
	20 to 28	24.2	24.2	24.3
Zones 13 to 15, over 1.70 sp. gr..	6 to 8	34.5	35.9	45.0
	8 to 10	33.5	37.0	43.8
Table feed, under 1.38 sp. gr....	10 to 14	6.9	6.9	7.0
	14 to 20	6.8	6.8	7.0
	20 to 28	6.8	6.8	7.0

[a] Moisture-free basis.

[b] A different table test from that reported but on the same feed.

The general trend exhibited by all the specific-gravity components of the raw coal—coal, light bone, heavy bone and refuse—may be summarized as follows: With the exception of the heavy bone, cubical shapes of the same specific gravity and screen size are discharged in greater proportion than are prismatic and flat shapes. The most obvious reason for the discharge of cubical shapes ahead of the more

flaky material is that in stratifying behind the riffles the flakes occupy a position lower in the strata than the cubes. In this position the cubes are subjected to higher velocities of cross-flowing water than are the flakes, and in addition they (the cubes) are less affected by the travel-inducing force supplied by the head motion, because they are farther from the surface of the deck. The general effect of this greater travel of flakes results in the discharge, at a given point, of flat material of a low specific gravity together with cubical material of a higher specific gravity.

The ash contents of cubical, prismatic and flat particles are of importance in discussing the relative travel and discharge of these shapes because of the direct relationship between ash content and specific gravity. Table 9 shows the ash contents of each of these shapes in several sizes of the feed and two of the zonal cuts. The ash contents of the three shapes of coal in zones 3 and 4 for each size are all within limits of allowable error, but the tendency is for the flakes to be slightly higher in ash than the cubes. The differences in specific gravity are, therefore, very small; hence it is probable that the greater travel of flat particles of coarse coal is due to shape rather than to specific gravity. It has already been pointed out in the earlier part of this report that the mass of a flat particle is only about one-half that of a cubical particle of the same square-mesh size. Table 9 also shows the ash contents of several sizes of the coal in zones 13 to 15. The coal discharged in these zones quite naturally consists of material considerably higher in specific gravity than the coal discharged in the earlier zones. Here the cubical particles show definitely higher ash contents than the prismatic and flat particles. Flat particles of coal of low specific gravity are accompanied by cubical coal of a higher specific gravity. All other specific-gravity fractions in zones 13 to 15 show higher ash contents in the flat material than in the prismatic and cubical shapes.

Summary and Conclusions

The report describes a method of screening in which particles are separated into three shapes—cubical, prismatic and flat—by means of two sets of sieves, one with square and the other with rectangular or slotted openings. The application of the method to the determination of shapes of particles in the specific-gravity components of a raw coal is shown. The proportion of cubical particles decreases and the proportion of flat particles increases with increase in specific gravity of the components of the raw coal examined. Numerical expression is given to the more flaky character of bone and refuse—a commonly observed fact.

The screening method was also used to study the influence of shape of particles in the treatment of a coal on a coal-washing table. The data show that cubical shapes in the coarse sizes are discharged first, and that flaky shapes of the same specific gravity and square-mesh screen size

are carried farther out on the table deck before they are discharged. In the fine sizes this condition is reversed; flaky material is discharged in greater proportion than cubical material. The predominating tendency, cubes ahead of flakes or flakes ahead of cubes, may depend upon the proportion of coarse and fine sizes in the table feed. The coal used in the work described consisted mainly of coarse sizes and showed cubes to be discharged first.

The tendencies revealed by this investigation have been discussed in some detail in an effort to analyze the influence of shape. The discussion may magnify the importance of shape. In general it must be concluded that shape of particle is a factor of minor importance in tabling this unsized coal, in so far as the over-all efficiency of the process is concerned. Size and, of course, specific-gravity difference are the major factors.

Acknowledgment

The author acknowledges his indebtedness to Messrs. B. M. Bird and W. H. Coghill.

DISCUSSION

(*Samuel A. Taylor presiding*)

P. Sterling, Wilkes-Barre, Pa. (written discussion).—I do not doubt that the size and shape of particles have a great influence on washing, but it does not seem to have sufficient importance from a practical standpoint of washing anthracite rice and barley coal. The former is $-5/16$ and $+3/16$, and the latter $-3/16$ and $+1/16$ or $3/32$. The necessity of additional equipment for presizing probably would more than offset the advantage of a slight gain in recovery.

Control of the Quality of Shipped Coal

By R. G. Baughman,* Linton, Ind.

(Pittsburgh Meeting, September, 1930)

With the constantly increasing sales competition, coal to be sold today must meet the test of quality in every respect. The producers must be able to make all marketable sizes that will meet such general requirements as follows:

1. Ash content must be consistently low, depending upon the use to be made of the coal.
2. Sulfur content must be controlled accurately according to requirements.
3. All sizes must be positively free from visible impurities.
4. Free moisture of washed sizes must be reduced to a minimum.
5. The freezing of wet coal in cars must be prevented.
6. The percentage of fines in small mixed sizes must be controlled to give the uniform and proper combustion when used on various types of automatic stokers.
7. The undersize particles or degradation caused by handling must be kept to an absolute minimum in all sizes loaded.

Many difficulties may be met in bringing any particular coal to meet these quality requirements, because the extent to which the quality of a coal may be controlled is determined largely by its physical characteristics.

Preparation of an Impure Coal Bed

The application of modern and scientific methods in the controlled preparation of bituminous coal from a bed in the mid-western field will illustrate the results that can be accomplished by adequate preparation:

The bed referred to contains coal inherently excellent as to low ash and other desirable characteristics; in fact, the coal itself is one of the best in the field. It is, however, divided into four layers by dirt bands of soft shale 1 to 2 in. thick. It also contains much sulfur in the form of sulfur lenses and laminated sheets. Horsebacks or rolls varying from 6 in. to 12 ft. in width are encountered in the mining of the bed and heretofore have rendered it practically unworkable. The mines operating in this

* Chief Engineer, Central Indiana Coal Co.

bed, heretofore almost without exception, have been unsuccessful because of the irregularity of the quality of the product.

Stripping and Preliminary Separation

The mine to be described is a surface or strip operation with which everybody is familiar in a general way. The coal is loaded by an electric shovel with a 3½-yd. dipper. The only effort made towards cleaning the coal in the pit is rough scraping of the surface of the bed with a caterpillar tractor equipped with a bulldozer blade, to remove the loose dirt and rock which accumulates from the stripping operation. The coal is then loaded into 35-ton, drop-bottom pit cars and carried to the tipple. At the tipple the cars are dumped automatically into a 250-ton concrete hopper. The capacity of the hopper is sufficient to act, to some extent, as a balance between the tipple operation and delays that may occur in the pit, thus giving a fairly continuous feed of raw coal to the washery, which is essential in any washing method. The run of mine coal is fed from the hopper by a reciprocating feeder into a 48-in. scraper flight conveyor with a capacity of 400 tons per hour. The feeder is driven by a variable-speed motor, of which the control box is placed on the washery floor. This gives the washery operator an opportunity to slow down the raw coal feed if a particularly dirty trip of coal has been delivered to the hopper and enables him to maintain a uniformly clean product.

Raw coal under 4 in. in size is screened out and delivered to the washer, while the 4 by 6-in. and 6-in. lump go to picking tables where impurities are removed by hand.

The rejects from the picking tables that contain sufficient coal to warrant their reclaiming are allowed to pass to a two-roll crusher, where the material is crushed down to $-2\frac{1}{2}$ in., thus freeing most of the impurities from the good coal. This product is run into the raw coal feed going to the washery.

The Washing Machine

For the purpose of separating the impurities from coal of -4 in., two units of the Simon-Carves system are used. Each unit has a capacity of 120 tons per hour of raw coal, which is first washed and later classified. This system is a comparatively new piece of equipment in this country. It consists of an all-steel wash box divided into five compartments. It operates on the principle of a jig, using air pressure as a pulsating medium instead of plungers. About 1¾ lb. of air is admitted to the surface of the water in each compartment by means of a piston air valve, operating at proper frequencies to give a continuous wave action over the screen bed. This system is simple in operation, has few moving parts, and requires little maintenance. One of the great advan-

tages is that both the air intensity and quantity of water needed may be controlled without interrupting the operation of the wash box. This ease of control is desirable in washing dirty coal because of the wide variation in the nature and amount of impurities.

Classifying, Washing and Drying

The clean coal from the wash boxes is carried in a launder to the classifying screens, where it is sprayed and rinsed with clean water to remove any sediment collected from the wash water. The various sizes are taken from the screens by spiral chutes to the loading booms or mixing conveyor, except the $-3/4$ in., which passes to dewatering screens. These screens are jigging trays fitted with wedge-wire screens having a $1/2$-mm. opening. After passing over these screens the fines are moderately dry and are then mixed with the other sizes as wanted.

Sampling Washed Coal

A Delatester is a great aid in controlling the quality of washed sizes, testing samples taken from each car while it is being loaded. The samples collected by cutting the stream of coal at intervals as it falls into the car are tested at once, which shows at all times the quality of the washed sizes as to ash content. If the tests show as much as 9 per cent. ash, the washery operator is notified immediately and the action of the wash box is corrected to bring the product to the proper requirements.

Float and Sink Tests

The solution used in making tests is maintained at 1.42 sp. gr. This figure was determined by the usual method of determining the specific gravity suited to a particular coal, as follows: A representative sample of the product to be tested is subjected to a series of float and sink tests at varying specific gravities. It is tested first in the heaviest solution of the series, say 1.60 sp. gr., then the float from that test is tested in the next lighter solution, say 1.58, and so on, using the float of the previous test in the next solution varying 0.02 lighter. At some point in the series there will be found a place where for two or three adjacent gravities no sink will be obtained. The mean of these particular gravities should give the proper gravity for operating the Delatester, and prevent any likelihood of having a material of the same gravity as that of the liquid used.

For the purpose of indicating the ash content a curve has been plotted (Fig. 1) from ash analysis made from samples of 100 per cent. float to 80 per cent. float on a solution of 1.42 specific gravity.

A complete daily report is made by a regular inspector, showing the quality for each car loaded—size, color and degree of cleaning. For all

washed sizes, the ash content indicated by the test made for each car is listed on the report. Copies of this report go to the operating and sales departments, which aids greatly in knowing day by day the quality of the product to be shipped. It also assists in tracing the cause or basis for complaint regarding any particular car.

Refuse and Coal Loss

The loss in refuse from the washery is about 15.5 per cent., of which 5 per cent. is float. However, 50 per cent. of the float in the refuse will pass a No. 10 screen and about 25 per cent. will pass a No. 100 screen.

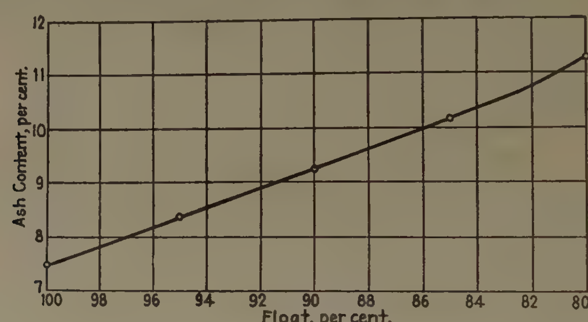

FIG. 1.—ASH CONTENT PLOTTED FROM ASH ANALYSIS OF SAMPLES OF FLOAT ON A SOLUTION OF 1.42 SPECIFIC GRAVITY.

It is desirable to keep the percentage of fines in the mixed washed sizes as low as possible, because of their ability to retain moisture. For this reason we might well consider the percentage of merchantable coal lost in the refuse as much less than 5 per cent.

The percentage of refuse from the hand-picked coal runs about the same as that from the washery. Naturally, this amount is large, because no attempt is made to clean the coal in the pit before it is loaded.

RESULTS OF THREE MONTHS' WORK

Table 1 gives the results obtained from the washery for the first three months of 1930.

TABLE 1.—*Results of Three Months' Operation in* 1930

Month	Coal Washed, Tons	Coal Graded No. 1, Per Cent.	Refuse, Tons	Refuse from Washer, Per Cent.	Float in Refuse, 1.42Sp.Gr., Per Cent.	Average Ash in Washed Coal, Per Cent.
January	23,856	100	6,480	15.52	5.10	8.34
February	19,171	100	4,356	15.15	4.95	8.33
March	16,763	100	4,390	15.53	5.00	8.34
Total and average	59,789	100	15,226	15.40	5.02	8.34

The result obtained with the complete preparation plant described has rendered it possible not only to market successfully the coal bed which had a "distinct black eye" on account of what is considered inevitable dirt, but through the efficacy of modern equipment, to obtain complete control of the product so that the shipper knows at all times the exact quality of his product. He has turned what used to be 18 to 20 per cent. ash screenings to screenings running below 9 per cent. in ash. The net result has been that the bed formerly regarded as unworkable has been transformed into a product that is now commanding a premium price over other coal in the same district—all accomplished by modern preparation.

Evaluation of Coal for Blast-furnace Coke

(A Review of the Literature)

By J. R. Campbell,* Scottdale, Pa.

(Pittsburgh Meeting, September, 1930)

It is the purpose of this paper to review somewhat in detail the literature on the subject that is extant, which ought to provoke considerable beneficial discussion.

The value of 1 per cent. ash in blast-furnace practice was brought to the attention of the writer when he was connected with the U. S. Steel Corpn. Various groups of blast furnaces in the Pittsburgh district compiled a statement in which the average saving was about 6 c. per ton on pig iron for every 1 per cent. of ash reduction. Later, the same groups of furnaces, when materials had reached a higher price, compiled a statement in which there was a 30-cent saving on pig iron for every 4 per cent. ash reduction in the Pittsburgh district, which was equivalent to 7½ c. for every 1 per cent. ash reduction. In the Chicago district, on long freight hauls of coal and coke from the Pocahontas district, the set-up was 30 c. per ton of pig iron for every 2 per cent. ash reduction, or 15 c. for every 1 per cent.

Some years ago, Ralph H. Sweetser, of the American Rolling Mill Co., presented a paper which provoked considerable comment.[1] To get the matter again before this body a résumé of his computations is given below:

Comparison of Pig Iron Costs

Ash in Coal, Per Cent.	Ash in Coke, Per Cent.	Fixed Carbon in Coke, Per Cent.	Coke per Ton of Pig Iron, Lb.	Pig Iron per Day, Tons
6.00	8.70	89.80	1800	365.0
7.00	10.15	88.35	1860	353.2

Coke with 8.70 *Per Cent. Ash*

Ore mixture, 4144 lb. = 1.85 tons @ $5.50	$10.17
Coke, 1800 lb. = 0.90 ton @ $6.50	5.85
Limestone, 1000 lb. = 0.50 ton @ $1.50	0.75
Labor, $365 per day on 365 tons	1.00
Supplies and service, $250 per day	0.68
Overhead, $200 per day	0.55
Reserve for relining and depreciation	1.00
Total	$20.00

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* Chairman, Committee on Evaluation of Coal for Blast-furnace Coke, Coal Division, American Institute of Mining and Metallurgical Engineers.

[1] R. H. Sweetser: Clean Coal—How to Get It, III. *Iron Tr. Rev.* (1925) **76**, 1445.

Coke with 10.1 *Per Cent. Ash*

Ore mixture, 4144 lb. = 1.85 tons @ $5.00	$10.17
Coke, 1860 lb. = 0.93 ton @ $6.50	6.05
Stone, 1034 lb. = 0.517 ton @ $1.50	0.77
Labor, $365 per day for 353.2 tons	1.03
Supplies and service, $250 per day	0.71
Overhead, $200 per day	0.57
Reserves for relining and depreciation	1.00
Total cost with 10.15 per cent. ash	20.30
Total cost with 8.70 per cent. ash	20.00
Increased cost	$ 0.30

In the foregoing statement it is indicated that 1 per cent. coal ash is equivalent to a saving of 30 c. per ton of pig iron.[2] About the same time that Mr. Sweetser wrote his paper, a reliable blast-furnace man in the Pittsburgh district calculated that 1 per cent. of ash was equivalent to 20 c. per ton of pig iron with coke at the same price as that Mr. Sweetser used.

To bring the subject more up to date, the writer within the last year has heard many comments on the value of 1 per cent. ash in blast-furnace practice. One operator figured that in his case the saving was 6.9 c. per ton of pig iron; a second, 16 c.; a third, 20 c.; and a fourth, 28 c. The average of these four statements is 17.7 c. per ton of pig iron for every 1 per cent. ash reduction.

Of course, conditions varied at each plant and to some extent depended upon the delivered cost or cost price received for slag and other materials, although the value of the alumina in the coke ash was somewhat important in some districts where the furnaces used iron or other mixtures lacking in this particular element. In fact, there are a great many variable factors which lead to many conclusions, but it seems to be a well-established fact that there *is* a substantial saving in blast-furnace practice for each 1 per cent. lower ash in the coke.

It is interesting to observe that the average given agrees almost exactly with the conclusions of a large steel concern in the Pittsburgh district that has set 17 c. per ton of pig iron for each 1 per cent. of ash reduction. The writer is inclined to the opinion that 15, 16 or 17 c. per ton is about the right figure to use in the Pittsburgh district.

This, of course, applies where the sulfur in the metallurgical coal is beyond the metallurgical limit. Below the metallurgical limits, the value of sulfur probably vanishes. Just what that metallurgical limit is, is somewhat of a question in this district and in other districts, but we may say that 1.25 per cent. sulfur is fair in metallurgical coal. It is necessary to have lower sulfur than this for the making of high-class foundry coke. He is very fortunate, indeed, who is able to produce sulfur of 1 per cent. or under in blast-furnace or foundry coke.

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[2] Another paper by Mr. Sweetser (One Per Cent. of Ash in a Ton of Coal) appeared in *Min. & Met.* (1924) **5**, 172.

At the February, 1930, meeting of the Institute, F. A. Jordan presented a progress report of the Committee, which was printed as *Technical Publication* 336. Mr. Jordan sets up an entirely different basis for the evaluation of coal for coke-making purposes. Instead of putting the entire saving on an ash basis, he splits the saving into ash and heavy-gravity material. The following is quoted from his paper:

> It seems reasonable that a coal of lower ash content and a small amount of high-gravity material should command a premium in addition to the base price Pittsburgh. That premium would be so many cents per unit. It is proposed for the consideration of the industry that a fair value per unit would be 3 c. for each per cent. of reduction in ash content and 4 c. for each per cent. reduction by weight of the high-gravity material. This plan of evaluating can possibly be better explained by an example. Assume a coal which by sampling and laboratory tests has been found to contain 7.2 per cent. ash and 1 per cent. by weight of high-gravity material. Assume further that trade has established a base price Pittsburgh of $3.12 per ton. This particular coal is entitled to a premium of 2.3 units at 3 c. per unit for ash reduction, or 6.9 c., and to a premium of four units at 4 c. per unit for reduction in the weight of the high-gravity material, or 16 c., making a total premium of 22.9 c. and thus evaluating the coal at $3.12 plus $0.229 or $3.349 per ton.

Mr. Jordan's base for the Pittsburgh seam of coal is 9.5 per cent. ash and 5 per cent. heavy-gravity material, run of mine.

This seems to be a sensible way of evaluating coal for coke-making purposes, as the modern coke-oven operator has come to believe that the influence of high-gravity material on coke structure is important, especially in the finer sizes, 0 by ⅜ in. We know of one large concern that has been willing to spend considerable extra money for the elimination of heavy-gravity material in the finer sizes on account of its deleterious effect on coke structure.

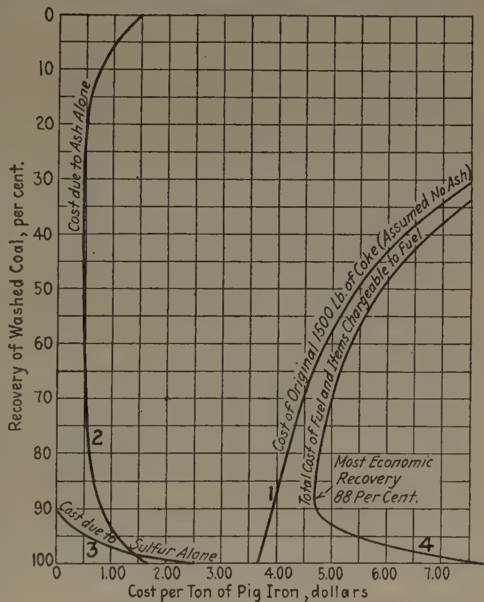
The greatest criticism that has been offered Mr. Jordan's set-up is that he has not made sufficient allowance for either the ash or heavy-gravity material. It seems to be the consensus of opinion that the 3 c. for ash reduction ought to be raised to 4 c. and the 4 c. for heavy-gravity material ought to be raised to 5 c. This allowance would take care of a reasonable preparation charge and be an inducement for the coal operator to get into mechanical preparation. Furthermore, such a basis would seem to be justified in blast-furnace practice if the foregoing figures have any significance.

Mr. Jordan's paper provoked considerable discussion, which is printed with the paper. The present writer, at the time, pointed out that rather intangible figures have been floating around for a good many years and that some responsible coal operator and blast-furnace man ought to realize these paper profits in practice. It seems high time for someone to try to put into practice an evaluation scheme like Mr. Jordan's, on a fair and equitable basis.

Economics of Sulfur in Coke

In the foregoing discussion no mention has been made of the savings in blast-furnace practice by the elimination of sulfur from metallurgical coal. This problem is involved, but it may be said at this time that every 0.01 per cent. of sulfur reduction is worth one penny per ton on pig iron in blast-furnace practice; thus, 0.1 per cent. reduction should be equivalent to 10 c. per ton on pig iron.

At the February, 1929, meeting of the Institute, George S. Scott, a chemical engineer, read a paper on Coal Washability Tests as a Guide to the Economic Limit of Coal Washing, which was issued as *Technical Publication* 159.[3] He tied in washability studies with the blast-furnace data. This point is particularly applicable where captive mines are operated in connection with steel plants. Mr. Scott's paper is comprehensive and merits the attention of all those interested in the subject. The final graph shown by Mr. Scott is appended. The most economical recovery curve takes a parabolic shape and shows that the economical point is 88 per cent. recovery of washed coal with 4.40 per cent. ash and 0.86 per cent. sulfur on the coal that was used for this particular purpose. This is a point that must be watched closely in inter-company business.



Summary of cost data.

Curve 1. Cost of blast-furnace fuel per ton pig iron at various washed-coal recoveries, assuming coke ash-free and sulfur-free.

Curve 2. Additional cost due to presence of ash.

Curve 3. Additional cost due to presence of sulfur.

Curve 4. Combined effect of all factors.

That coal is not the Pittsburgh seam of coal. It is coal from one of the lower productive measures. This is very unfortunate, as it should have been the Pittsburgh seam of coal for the purposes of this paper.

Discussion on Ash and Sulfur Desired

In the absence of any specific paper on the subject of ash and sulfur, we hope that this recapitulation of the most outstanding points in the

[3] G. S. Scott: *Trans.* A. I. M. E., Coal Div. (1930) 287.

literature will provoke discussion among the members at this time and give us all food for thought.

Your Chairman is making a strong effort to get the subject before the coke-oven and blast-furnace operators as well as the coal operators. They must work hand in hand and agree on a fair and equitable basis, otherwise the work done in preparation by the coal operators will not receive the benefits due it; neither will the blast-furnace man profit unless he is willing to split the profits with the coal operators.

We are all agreed on the fundamentals and it only remains for the coal operator and blast-furnace man to get together.

DISCUSSION

(*J. R. Campbell presiding*)

J. R. Campbell, Washington, D. C.—Coal preparation in America has made great strides within the past year or two. Statistics show that the total amount of coal prepared has increased from 27,000,000 to a little over 37,000,000 tons annually in a little over one year. The total, on an annual production basis, of a little over 500,000,000 for last year means that about 7 per cent. of all the coal produced in the United States is prepared mechanically, either by wet cleaning or dry cleaning methods. I believe that the wet methods of preparation prevail in this country. Out of the 37,000,000 prepared by both methods, I think that the pneumatic method is credited with about 5,000,000 tons, although the increase in pneumatic methods has been rapid within the past year. I think the percentage of increase by air methods was a little more than the percentage of increase by the wet methods.

There is some fundamental reason why coal preparation is going forward by leaps and bounds in this country. Before the end of 1930, I expect to see 45,000,000 tons of coal prepared in the United States, perhaps 50,000,000 tons, or 10 per cent. of the annual output.

R. H. Sweetser, Columbus, Ohio (written discussion).—Mr. Campbell has given a concise summary of the rather meager literature on the value of 1 per cent. of ash in coking coals, but he has introduced terms that are misleading. The expressions are as follows, in the order that they occur in his paper.

1. "The value of 1 per cent. ash in blast-furnace practice."
2. "Every 1 per cent. of ash reduction."
3. "Every 4 per cent. ash reduction in the Pittsburgh district."
4. "For every 1 per cent. ash reduction."
5. "Value of 1 per cent. ash in blast-furnace practice."
6. "Every 1 per cent. ash reduction."
7. "There is a substantial saving in blast-furnace practice for each 1 per cent. lower ash in the coke."
8. "For each 1 per cent. of ash reduction."

Judging from expression 7, the "1 per cent. of ash reduction" under discussion is a 1 per cent. reduction of the ash in the blast-furnace coke. Such a meaning is entirely different from the basis of my calculations, which have been published on the subject of the evaluation of coal for blast-furnace coke. Lest there should be confusion in making these comparisons, let me repeat that the "1 per cent. of ash" under consideration is in the *coal*, and not in the coke. The calculations quoted by Mr. Campbell in his paper were first worked out from actual cost sheets and actual coal, coke and blast-furnace records passing over my desk early in 1922. They were published on June 4, 1925, in the *Iron Trade Review*.

At the annual meeting in New York in February, 1926, I presented the question of evaluation of coking coals to the Institute and gave statements to show the reduction in pig-iron costs when the ash in the coals charged into the by-product coke ovens is reduced. Obviously the gain in lower pig-iron costs should not all be retained by the pig iron; a part of this gain rightfully belongs to the coking coal, whether that coal be natural coal direct from the company's own mine or purchased coal, or coal that has been beneficiated by some mechanical cleaning process. Just how much total value should be placed on each 1 per cent. of ash reduction depends on several varying factors, all of which must be adjudicated by conference and agreement after obtaining as many facts as possible.

1. The consequential reduction in the cost of pig iron by reducing the percentage of ash in the coking coal depends on the percentage of volatile matter in the coal mixture that is to be coked, because the higher the volatile matter, the more pounds of coal will be required to produce 1 ton of furnace coke, and the factor that determines the percentage of ash in the coke will be correspondingly greater. This is shown in Table 1.

2. The temperature at which the coal is coked will have a direct bearing on the increase of the percentage of ash in the coke over the percentage of ash in the coal. It is possible to have an increase of fixed carbon by deposition of carbon other than the theoretical fixed carbon in the coal (Table 1).

3. The degree of pulverization of the coal mixture, and any other factor that influences the percentage of coke breeze, will directly affect the increase of ash in the coke over the percentage of ash in the coal (Table 1).

There may be other factors known to coke-oven operators, but these three items will serve to show that the evaluation of 1 per cent. of ash in a ton of coking coal is a very complicated calculation. But it can be done. The facts given in Table 1 show how it is possible for Mr. Campbell to obtain such widely varying figures as to the extra cost of pig iron for an increase of 1 per cent. in the ash in a coking coal.

Table 1 is taken from actual reports of by-product coke-oven operations. The two months were chosen because the difference in the percentage of ash in the two

TABLE 1.—*Reports of By-product Coke-oven Operations*

	May, 1927	October, 1925
Volatile matter in coal, per cent	32.35	31.36
Ash in coal, per cent	5.94	6.93
Tons coal per ton coke	1.440	1.416
Ash in coke, per cent	8.26	9.36
Fixed carbon in coke, per cent	90.07	89.57
Sulfur in coal, per cent	0.72	0.78
Sulfur in coke, per cent	0.58	0.63
Through 1/8 in., per cent	70.87	72.17
Coking time, hr	28.94	19.32
Theoretical yield, per cent	68.70	70.90
Actual yield, per cent	69.42	73.26
Gain in yield, per cent	0.72	2.46
High-volatile coal, per cent	91.00	87.00
Pocahontas coal, per cent	9.00	13.00
Moisture in coal, per cent	3.99	3.80
Moisture in coke,[a] per cent	2.36	3.65
Breeze per ton coke, lb	114	126

[a] Shipping weights of coke adjusted to 3 per cent. moisture whenever moisture exceeds 3 per cent.

coal mixtures was almost the same as I used in my original calculations 9 years ago. The figures shown are averages for the 31 days of operation in each month.

I went over a great many records and found that the coal put into the coke ovens in May, 1927, was about 0.99 per cent. different from what went in in October, 1925. The ash in the May coal was 5.94; in October it was 6.93. In May it required 1.440 tons of coal to make 1 ton of furnace coke—blast furnace coke—and in October it took 1.416. The ash in the May coke was 8.26 and in the October coke was 9.36. The fixed carbon in the May coke was 90.07, and in the October coke it was 89.57. The sulfur in the May coal was 0.72 and in the May coke it was 0.58. The sulfur in the October coal was 0.78 and in the coke it was 0.63. There is a big difference in the coking time. In May the coking time was 28.94 hr. That was a very slow operation, and the ovens were kept cool; the volatile in the coal mixture was 32.53 per cent.

Mr. Campbell spoke about the effect of "high-gravity material." I do not like that name. If he will call it "extraneous ash" it will be better, because some of our coals are of much higher gravity than other coals, and we might be confused, but if he speaks of "extraneous ash," we know that is slate and stone and everything like that.

Through ⅛-in. mesh in May there was 70.87 per cent. of the coal, and in October, 72.17. The grinding was not quite so fine. In the May mixture there was 91 per cent. of high volatile and 9 per cent. of Pocahontas coal. In October there was 87 per cent. of high-volatile and 13 per cent. Pocahontas. The breeze in May was only 114 lb. per ton of coke and in October it was 126, the higher ash coal showing a higher amount of breeze.

In the May coke the moisture was only 2.36 per cent. In the October coke it was 3.65 per cent., but when that coke went to the blast furnace, the moisture was adjusted to 3 per cent., so that in Table 2, in which we show the consumption of the coke, it was based on coke with 3.24 per cent. moisture, but the cost and the weights were adjusted to 3 per cent.

My original figures, quoted by Mr. Campbell in his paper, show an increase of 30¢ in the cost per ton of pig iron, when there was an increase of 1 per cent. in the ash in the coking coal used to make the coke that was costing $6.50 per ton of coke delivered at the blast furnace. Of this increased cost, 20¢, or two-thirds, was in the 60 lb. extra coke required; 2¢ was in the extra limestone; and the balance, 8¢, was on account of the decrease in production of 11.8 tons pig iron per 24 hr., and the consequent increased per ton cost of labor, supplies, service and overhead (including office expense, taxes, insurance). That is, the higher ash coke actually gave a lower production by 11.9 tons of pig iron, and that increased the cost per ton of labor, supplies and overhead.

The records and cost sheets of coal, coke and pig iron that I have studied since 1922 have strongly confirmed my conclusions and estimates of the extra cost for each 1 per cent. increase in the ash in the coking coal. Mr. Campbell relates that "a reliable blast-furnace man in the Pittsburgh district calculated that 1 per cent. of ash was equivalent to 20¢ per ton of pig iron with coke at the same price as that Mr. Sweetser used." It would be interesting to have all his itemized costs the same as shown in my calculations, where there were no credits for slag, gas or flue dust. The operator who figured a saving of only 6.9¢ per ton of pig iron "for 1 per cent. ash in blast-furnace practice" was certainly not using the same kind of chemistry and track scales that I used, nor the same kind of cost accounting. You see, there is a confusion. This paper does not tell whether that 1.0 per cent. was a reduction in ash in coal or in coke. I am just quoting Mr. Campbell that it was "1.0 per cent. ash in blast-furnace practice."

The whole iron and steel industry is based on a unit of iron and a unit of heat.

A coal that has 1 per cent. more ash than another coking coal will make a coke with 1.00 to 1.50 per cent. more ash than the ash in the second coal, depending on the

volatile matter and coking time, which are supposed to be the same in both cases in my calculation. The coke with the higher ash will have less heat value than the other coke, other things, such as volatile matter, moisture and structure, being the same. The more ash, the less fixed carbon, and the more coke required per ton of pig iron.

To get comparable records of blast-furnace operations where a whole month's performance can be compared with that of another month is very difficult, especially when such variables as coking-coal analysis and coal mixture and coke-oven operations are involved in the comparison.

In the calculation quoted, the blast-furnace results with 6.00 per cent. ash were compared with a coal with 7.00 per cent. ash. The two cokes had 89.80 and 88.35 per cent. fixed carbon respectively, and the coke per ton of pig iron was 1800 lb. and 1860 lb. respectively. The ore mixture per ton of iron was 4144 lb. in both cases.

A search through many cost sheets was made to find actual results where there was 1 per cent. difference in the ash in the coking coals. The nearest I could find was for one of the blast-furnaces of The American Rolling Mill Co. using identically the same weight of ore mixture per ton of pig iron for the two months compared, and using cokes that were made from coals having ash contents 0.99 per cent. apart.

The two months of blast-furnace operation to be compared are the two months of coke-oven operation, May, 1927, and October, 1925. The quantities of ore, coke and limestone used are the actual weights taken from our cost sheets. The daily cost for labor, supplies and service, and for overhead are the actual costs for one of the months and the same daily total costs were used in getting the comparison for difference in daily tonnage. It happened that the tonnage record in October, 1925, was a new high record, and again in May, 1927, it was a new high record for the same blast furnace.

Just why the furnace made iron with 101 lb. less coke in May, 1927, than in October, 1925, I cannot yet see, unless it should be in the analysis of the slag; this difference is greater than can be calculated by any known rules. It is a significant fact, however, that the average daily tonnage of coke burned in October, 1925, was 359.47 tons, and in May, 1927, it was 360.23 tons. The rated capacity of this furnace is 340.2 tons coke per 24 hr., so that the furnace was driven at 105.66 per cent. and 105.88 per cent. of rated capacity during the two months being compared. In the month of lower coke there was lower blast temperature, higher humidity and higher temperature of the outdoor air, higher silicon and higher sulfur in the pig iron. I doubt whether the 1.12 per cent. higher fixed carbon in the May, 1927, coke would account for as much as one-half of the 101 lb. of coke, but I do believe that the lower melting temperature of May, 1927, slag helped to reduce the coke per ton of pig iron. I also recognize the probability of intangible (at present) differences in the characteristics of the coke.

The salient facts of the blast-furnace operation are shown in Table 2, which is the continuation of Table 1 in that it shows the results in the blast furnace using the cokes made from the coals described in Table 1.

The actual cost of the pig iron in May, 1927, was so much less than in October, 1925, all costs based on the same prices, that I shall not state the decrease because of the possibilities of misunderstanding, but the decrease was more than the 30 c. given in the calculations quoted by Mr. Campbell. The saving in coke cost was 61.75 per cent. of the total saving; the saving in limestone cost was 16.41 per cent. of the total saving; and the balance, 21.84 per cent., of the decrease in cost of pig iron, was brought about by the increased daily tonnage. According to Mr. Campbell's remark on the "economics of sulfur in coke" there should be a credit of 5 c. for the 0.58 per cent. sulfur in the May, 1927, coke compared with the 0.63 per cent. sulfur in the October, 1925, coke. I know that the sulfur has something to do with coke consumption, but do not have quantitative data.

TABLE 2.—*Results at Blast Furnace Using Cokes Made from Coals Described in Table* 1

	May, 1927	October, 1925
Ash in coal, per cent	5.94	6.93
Coke per ton pig iron, lb.	1632	1733
Stone per ton pig iron, lb.	575	714
Ore per ton pig iron, lb.	4046	4046
Pig iron per day, tons	441.4	414.8
Silicon in pig iron, per cent	1.09	0.89
Sulfur in pig iron, per cent	0.029	0.026
Blast temperature, deg. F.	1018	1125
Outdoor temperature, deg. F.	61	47
Moisture, grains	4.66	3.24
Flue dust, daily, tons	27.3	19.8
Silica in slag, per cent	36.13	34.76
Theoretical yield, per cent	57.24	57.13

To further substantiate the statement that 1 per cent. of ash in the coking coal adds to the cost of the pig iron made with that coke, I give in Table 3 a part of the results of an actual test of coal, coke and pig iron in which a coal mine, a by-product

TABLE 3.—*Portions of a Test to Determine Relative Values of a Coal Mixture*

	16 Days	14 Days
Coal Mines		
Average ash in coal mixture, per cent	7.01	6.81
Average volatile in coal mixture, per cent	33.67	32.06
Average sulfur in coal mixture, per cent	0.85	0.83
Average moisture in coal mixture, per cent	3.55	3.69
Coke Ovens		
High-volatile coal in coal mixture, per cent	80.00	75.00
Low-volatile coal in coal mixture, per cent	20.00	25.00
Average through ⅛-in., per cent	68.99	67.82
Average coking time, hr.	22.64	24.65
Average ash in coke, per cent	10.42	9.52
Average sulfur in coke, per cent	0.71	0.67
Average weight per cu. ft. of coke, lb.	29.4	29.9
Blast Furnace		
Average pig iron per day, tons	399.9	417.3
Average coke per ton pig iron, lb.	1755	1695
Average limestone per ton pig iron, lb.	845	858
Average ore per pound of coke, lb.	2.07	2.13
Average charges driven per 24 hr.	97	100
Air at 60° F. per pound coke, cu. ft.	57.2	57.4
Average blast pressure, lb.	14.22	13.98
Average blast temperature, deg. F.	1107	1114
Average temperature outdoor air, deg. F.	36	34
Average moisture per cu. ft. air, grains	2.39	2.10
Actual yield of metallic mixture, per cent	57.25	56.37
Average silicon in pig iron, per cent	0.96	0.90
Average sulfur in pig iron, per cent	0.031	0.027
Average silica in slag, per cent	35.95	35.85
Average sulfur in slag, per cent	1.05	.99

coke plant and a blast furnace were all working together for a month's test to determine the relative values of different coal mixtures. Although two weeks is hardly long enough to make a thorough test of the value of a blast-furnace coke, yet these two periods of 16 days and 14 days respectively were long enough to give sufficient data for drawing our conclusions. The 30-day test reported here was the middle part of a test that lasted from January 28 to March 6, 1927.

Here is a case when a decrease of 0.20 per cent. in the ash in the coking coal caused a decrease of 0.90 per cent. in the ash in the furnace coke, but there was also a decrease of 1.61 per cent. in the volatile matter in the coal mixture. There was also a change in the proportion of high-volatile and low-volatile coal in the coal mixture that went into the by-product coke ovens. The quality of the coke with 80 per cent. high-volatile coal was not as satisfactory as with 75 per cent., chiefly on account of the nature of the high-volatile coal, which contained 36.84 per cent. volatile matter and 7.71 per cent. ash.

There is no question as to the advantage of lower ash coals for blast-furnace coke. The old idea of having "ash in the coke so as to make a coke strong enough to support the burden" is obsolete, but there are men who not only believe it but act on it.

Only last month, in discussing coke, I found a man who actually had a coal mine that furnished him with coal with about 3.5 to 4.5 per cent. ash, and gave him a coke that was very low in ash, and he deliberately went outside and got high-ash coal to put in, so that he could get a 12 per cent. ash in his coke.

I believe that the strength of blast-furnace coke is independent of the ash contents. My experience with the biggest charcoal furnace in the world at Sault Ste. Marie, Ontario, in 1905 (No. 1 furnace The Algoma Steel Co.)[4] proved that charcoal could come to the tuyeres of a 70-ft. blast-furnace without being crushed. I have a piece of charcoal, 1⅜ by ⅞ by 1¼ in., that I picked out of a lump of slag at the site of Argillite furnace in Greenup County, Kentucky. This old charcoal furnace, only 25 ft. high, was built in 1818, and has not been in blast for probably over half a century; yet this piece of charcoal, which went down through the furnace and out with the roughing slag, and which has lain on the banks of the Little Sandy River all these years, is still a compact piece of good blast-furnace fuel. Even the rings of the years of the tree from which the charcoal was made can be seen.

At the time I found this piece of charcoal, I found also a piece of charcoal pig iron that contained 0.094 per cent. sulfur and 2.64 per cent. silicon. This high percentage of sulfur in charcoal iron seemed impossible because the charcoal pig iron made from Lake Superior ores is exceptionally low in sulfur in spite of the glassy slags. The old Kentucky iron ores must have carried considerable sulfur, although they were usually roasted before being put into the furnace.

Mr. Campbell has opened wide the door for a study of the behavior of sulfur in the hearth of the blast furnace. The fundamentals of the elimination of sulfur are being studied by the U. S. Bureau of Mines. Meanwhile, we all should be collecting operating data.

A discussion of Mr. Campbell's paper is not complete without further reference to F. A. Jordan's paper, the essence of which was quoted by Mr. Campbell. In my discussion of Mr. Jordan's paper last February, I expressed my disappointment that he had chosen the ash in the coal as the measure of evaluation instead of the carbon, but at that time I had not seen his paper, which was the first official statement of the committee on coal evaluation appointed nearly four years ago. After studying Mr. Jordan's paper and the criticisms that followed, I am convinced that he has proposed a sound foundation for a workable method for the evaluation of coal for

[4] R. H. Sweetser: Charcoal and Coke as Blast-furnace Fuels. *Trans.* A. I. M. E. (1908) **39**, 228.

cokemaking purposes, not only for blast-furnace use but for gas plants and for steel-plant producers. I would like to second his proposal for a Pittsburgh-seam base of 9.5 per cent. total ash, "which contains 5 per cent. by weight of +1.55 sp. gr. material when crushed to −2-in. round-hole mesh."

Along the main lines of the Chesapeake & Ohio and the Norfolk & Western R. R., we speak of those two undesirables as "inherent ash" and "extraneous ash"; the two together make up the "total ash." We also speak of "inherent sulfur," which never leaves us until it passes out of the hearth of our blast furnaces either with the slag or with the iron. Like inherent ash, no coal-cleaning method can remove it.

K. C. Appleyard, Birtley, England.—One of the great companies in the north of England, which probably is more advanced in its coke-oven work and possibly in its steel preparation—that is, it has one of the most modern steel works, at least in Britain—has done a great deal of work on this matter, and my recollection is that it found as much financial advantage to be gained from the use of graded coke in blast furnace as in the use of clean coal, and while I would not like this to go into your record as a carefully authenticated fact, my recollection is that with a reduction of 4 per cent. of ash in the coke—not the coal this time—and the use of carefully graded coke, the output of its blast furnaces was increased by well over 50 per cent. To such an extent, in fact, that instead of doing as so many people have done, tearing down its prewar furnaces and putting up very elaborate and expensive furnaces to replace them it still has its prewar furnaces in use. And I believe it is getting 100 per cent. greater production out of those furnaces than before the war, by careful study of the coal, coke, and other matters.

The only figure that I have available here is one which indicates that a reduction of 1 per cent. in the ash content of the coke charged to the furnace reduces the coke consumption by 40 to 45 lb. per ton of pig iron. Knowing very little about it, I should be quite prepared to stand corrected on this matter, but those are the figures which I have available which I can rely upon.

Mr. Sweetser raised one point which is a subject of a good deal of interest to me—that ash in the coke does not increase its strength. That is an argument that I have heard thrashed out many times, and in a particular coke-oven plant which has arrangements for large-scale experimenting and the latest technical help of every kind I think it has been proved beyond any doubt that the addition of fine stone in the coal has undoubtedly cured the sponginess that is found in these particular coals. What the explanation of it is, I do not know, but the fact remains that the difference between one set of conditions and the other is unquestionably the condition between spongy coke and nonspongy coke, and I should very much like to get some enlightenment on this subject.

J. R. Campbell.—I think Mr. Sweetser's slant on the American practice is about the slant of the coal fraternity—that ash and coke in American coals do not increase the strength. There may be a few exceptions that I do not know about. Spongy coke has been improved by the addition of ash.

R. H. Sweetser.—I would like to make an explanation of the two months chosen for record in my written discussion. I said that it was very hard to get comparable records, and I chose those particular months because the amount of ore per ton of pig iron in the two months was identical. It was the same furnace, a 31-day month, making the same kind of basic iron. The blast temperatures were pretty nearly the same; 1018° F. in the month with the lowest coke and 1125° in the month with the highest coke, and so many of the conditions were identical that I chose those two months out of a great number of records.

In going over the records, I have not yet found that a higher ash has given a lower coke consumption, but I can get many records out of the cost sheets of several groups of blast furnaces where this same trend of lower ash and lower coke consumption holds true. So those two months have more in common than almost any two months I could find.

T. T. Read, New York, N. Y.—About the time that Mr. Sweetser was making calculations on coke, I was attempting the task of trying to evaluate the whole range of raw materials used in making a ton of pig iron. The blast-furnace people kindly provided me with the actual records of 26 furnaces for two consecutive months, so that I could calculate the quantities in pounds. On plotting the results I was confronted with the appalling fact that even after you take into account the silicon, carbon and so on in the pig iron, some of the furnaces were making a ton of pig iron with several hundred pounds of iron less than was required and some of the others were making it from several hundred pounds more than was necessary. Certainly a blast furnace cannot make iron out of air, nor cause it to evaporate. So there is no question that the original figures on weights and analyses were not sufficiently accurate to make possible such a calculation, and I abandoned the attempt. If the error in the iron, which was the ingredient on which you would expect the most accurate figures, was so large, it seemed useless to go on. We need more accurate and more figures on the subject before we can come down to any precise balancing of costs.

R. H. Sweetser.—Dr. Read is right, but all the records that I have presented here were made after having had under consideration a balance sheet of every unit of iron that went into the furnace during those months.

T. T. Read.—It is gratifying to learn that somebody is putting some money into this. People who are actually putting up some dollars and cents have a financial incentive to try to find out how much it does cost. I did not know, up to this time, that anybody was actually paying more for a better grade of coke. We are all agreed that the better quality of coke does cheapen the cost of a ton of pig iron. All we are arguing about is how much.

J. R. Campbell.—That is right. I might say that the big steel companies do not make many mistakes in going into beneficiation schemes. We have some very big groups of steel people in the country that have looked into this pretty carefully, and very lately are getting into the beneficiation of coal for coking purposes. In order to get appropriations from New York, or Chicago, or from other places, they have to get that in pretty good shape before they can get the money to put into the job. On the other hand, there may be conditions that are intolerable, forcing the operators to put in cleaning plants so that they can use the coal at all.

R. P. Hudson, Wayland, Ky. (written discussion).—The effect of an increase of 1 per cent. in the ash of coking coal on blast-furnace tonnage and production cost was first brought to my attention in a copy of a report addressed to an executive of a large steel company, in the spring of 1922. The report showed that when the ash in the coking-coal mixture increased from 6 to 7 per cent.—a 1 per cent. increase—the production of the blast furnace decreased to the extent of 11.2 tons per day, and there was a simultaneous increase in the cost of the iron produced, amounting to 27.5 c. per ton. In order to have the same pig-iron cost with coke made from the coal containing 7 per cent. ash as with the coke made from that containing 6 per cent. ash, there would have to be a decrease of 30 c. in the cost of each ton of coke delivered at the furnace.

The furnace on which these calculations were based had a capacity of approximately 365 tons of iron, using a straight ore mixture, exclusive of scrap or other iron-

bearing material. The fuel consumption was practically ideal, when the analyses of the raw materials were considered, and the cost of the delivered coke was $6.90 per ton. Of course, such calculations as were involved in the report were necessarily based almost wholly on theoretical considerations; but in this connection, I was informed that the figures arrived at were subsequently substantiated by actual operating data.

Mr. Campbell has given a résumé of Mr. Sweetser's calculations, as presented in a paper before the Institute several years ago. I believe that Mr. Sweetser's figures are based upon proper assumptions and that they are substantially correct. His figures show that a 1 per cent. increase in the ash of a coking-coal mixture causes an increase in the unit cost of coke, flux, labor, supplies and overhead. In fact, the unit cost of ore, and the relining and depreciation reserve, are the only two items in the cost sheet which are not affected. Mr. Campbell points out the coincidence that "about the same time that Mr. Sweetser wrote his paper, a reliable blast-furnace man in the Pittsburgh district calculated that 1 per cent. of ash was equivalent to 20 c. per ton of pig iron, with coke at the same price as that Mr. Sweetser used." It would be interesting to see a comparison of these two sets of figures. There must have been some differences in costs, other than fuel, and, it is probable that the Pittsburgh man's figures were based upon a furnace with a greater capacity than Mr. Sweetser's furnace. It should also be pointed out that Mr. Sweetser used a "base coal" containing 6 per cent. ash, and it is probable that the use of a base coal containing more or less ash than the one used would have produced more or less of a corresponding change in the results obtained. I believe that the coal and iron industries should adopt a hypothetical coal analysis, to serve as a base, just as the iron-ore industry adopted base ores a good many years ago.

The amount of coal necessary to produce a ton of coke is also important. Mr. Sweetser has assumed this to be 1.45 tons in each instance. If he had used 1.4 or 1.5 as the volume of coal necessary to produce a ton of coke his results would have been slightly different. He has assumed a volatile-matter content of 1.50 per cent. for each coke.

Turning now to sulfur; since both ash and sulfur are impurities in coking coal, they may simply be identified as such and their combined negative values estimated simultaneously.

In order to evaluate by-product coal, we should first establish a base coal analysis. It seems to be the general opinion of by-product coke-oven operators that the most economical coal to use in the ovens is one that requires 1.4 tons to produce 1 ton of coke. Theoretically, such a coal contains 28.57 per cent. volatile matter, on the assumption that the coke produced is free from volatile matter. Since recently blast-furnace operators have indicated that they desire coke with maximum ash and sulfur contents of 10 per cent. and 1 per cent. respectively, it is evident that our base coal may contain 7.14 per cent. ash and 1.43 per cent. sulfur. It is evident then that the fixed carbon content of the base coal is 64.29 per cent. Moisture has been disregarded in these calculations, but it should be taken into consideration when comparing the value of a given coal with the base coal.

By assuming the ash composition of all coals to be similar, I recently made some calculations which showed the theoretical negative value of 1 per cent. of ash in by-product coal. By using a base coal of the composition given above, and applying the calculations to a blast furnace with a capacity of 500 tons of pig iron, it was shown that a coal containing 4.50 per cent. less ash than the base coal was worth 57 c. more per ton to the blast-furnace operator. This is equivalent to 12.66 c. per 1 per cent. reduction in ash, and the decrease in the production cost of the pig iron for each 1 per cent. of ash reduction in the coal was 18.14 c. This figure agrees fairly well with the average of the four results quoted as originating in the Pittsburgh district, and

is only slightly higher than Mr. Campbell's opinion of what it should be in that region. On the other hand, the fact should be brought out that my cost of materials, etc., was lower than Mr. Sweetser's cost, while the furnace production was considerably greater. Mr. Sweetser evidently made use of Forsythe's rule for equating sulfur and ash to coke and stone, thus giving him a theoretical basis for calculating the cost, in terms of coke and stone, of increments of coke impurity. On the other hand, I established a standard limestone analysis and figured the burdens in detail. This may make some difference in the theoretical amount of available carbon and consequently the fuel consumption per ton of iron.

In another set of calculations, I estimated the relative values of the by-product coal originating from the various mines with which I am associated. A recapitulation showed that there was not an exactly corresponding increase in the value of the coal for each increment of ash reduction, but that when the results were averaged the quotient agreed rather well with the results arrived at in other computations. The average result was 14.16 c., as the negative value of 1 per cent. of ash. This is in fair agreement with the 12.66 c. arrived at in the former calculation, especially when we consider that corresponding blast-furnace capacities were not used. The average decrease in cost of pig iron for each 1 per cent. decrease in ash was exactly 20 c.

We may correlate the data and note that it agrees very well with the figures supplied by Mr. Campbell's paper. The average of all of the writer's results indicates that the production cost of pig iron may be decreased 19.07 c. by reducing the ash in the coking-coal mixture 1 per cent.; and when we reduce the ash in the coking coal 1 per cent., we increase its value to the extent of 13.4 c. per ton.

While we have all obtained somewhat different results on this problem, the fact is evident that there is a considerable negative value to 1 per cent. of ash in coking coal. The principle of the thing has been solved and generally agreed upon, and it seems natural to conclude that the details of the proposition can be adjusted on an equitable basis through the cooperation of the blast-furnace man and the coal operator.

Economic Utilization of Natural Gas

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(New York Meeting, February, 1931)

This paper presents the results of a study of the comparative values of the several fuels commonly used by industrial plants. It shows that the energy actually recovered from any fuel and turned into useful work is dependent upon several factors: (1) the total energy available in the fuel, (2) energy losses resulting from combustion reactions and (3) energy losses dependent upon the mechanics of the firing operation.

The chemistry of combustion and the losses resulting therefrom are discussed. The advantages and disadvantages of the several mechanical methods of firing and the resulting heat losses in each are developed. Data obtained from several typical fuel operations, where careful records have been maintained, are presented to show that the practical results in the actual operation are in harmony with the theoretical results developed in the earlier part of this paper.

Water power and solid, liquid or gaseous fuels are at present the only commercial sources of energy. The economic utilization of any one of them depends, in the final analysis, on the cost of energy per unit of finished product to which the energy is applied. The location of the plant utilizing the particular fuel chosen is a large factor in the original choice. The cost per unit at its original natural source, or even in its delivered form, is not always the final measure of its worth. The use of electricity as a source of energy illustrates this to the extreme. Electric current produced from water power is very cheap in certain localities. In Norway, current is sold as low as 1/20¢ per kw-hr., while in dry cells it costs as much as $10 per kw-hr. Yet the sale of dry cells is a large business. The selling price of gas also varies widely. In some localities natural gas for the manufacture of carbon black sells for as low as $0.02 per million British thermal units, while artificial gas for domestic purposes sells for as high as $6 per million B.t.u. Obviously the cost per unit of energy is not the final determining factor in its purchase.

Any discussion of the economic utilization of fuels must include costs in relation to the units of finished material manufactured. In large central power generating stations, this is the over-all cost per kilowatt-hour, and in ceramic firing the cost per unit of finished product of a certain quality is the criterion of economic utilization.

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The combustion of various fuels is the main source of energy in the world today. Many factors beside the cost per heat unit must be taken into consideration in the choice of the most economical fuel for a given operation. It is possible that a property of one fuel that makes it ideal for one operation may handicap it in another one. This is illustrated in cement burning. The finely divided ash of powdered coal is added to the bulk of the cement clinker, and gaseous or liquid fuels for this same operation are actually penalized for the lack of ash. To arrive at these various factors in determining the economics of the proper utilization of fuels we must first consider the mechanism of combustion.

Mechanism of Combustion of Fuels

Solid Fuels

Solid fuels are burned either on grates or in pulverized form. Coal burned on grates has a triple cycle in its combustion. The first step is the distillation of the volatile constituents; the second, their combustion. When the hydrocarbons which largely compose the volatile portion of coal are distilled they either combine with oxygen to form carbon dioxide and water or are decomposed thermally to form finely divided carbon and hydrogen. This carbon, if not burned, leaves the furnace as smoke and, of course, is a direct heat loss. The oxygen in the air draft underneath the fuel bed does not reach the zone of distillation in sufficient quantities to completely oxidize these hydrocarbons. It is necessary, therefore, to introduce secondary air above the fuel bed to accomplish this. With hand-fired boilers it is difficult to get the firemen to place the coal at the front of the fire box, so that the volatile matter may be distilled and burned before leaving the furnace. In stoker-fired furnaces the coal is progressively fired and more of the volatile hydrocarbons are consumed. The stoker also in a larger measure eliminates the human element. In the third step the residual carbon left in the coal after distillation begins to react with the oxygen of the draft when it reaches a temperature of about 1300° F. The primary reaction is $C + O_2 = CO_2$.

The rate of disappearance of the oxygen is almost exactly equal to the increasing carbon dioxide content of the draft. Near the surface of the fuel bed the oxygen content has dropped to 2 to 4 per cent. from about 21 per cent. in air. This condition does not change even at very high air velocities and all evidence seems to indicate that the rate of combustion is controlled by the rate of diffusion of the oxygen through the surface films around each carbon particle.

Another reaction that takes place is $CO_2 + C = 2CO$. This reaction is controlled by the rate of chemical action of carbon dioxide and carbon. It increases with temperature but even at high temperatures it is slower than the carbon and oxygen reaction to give CO_2. Hence the necessity

for having deep fuel beds to allow long contact between the oxygen and carbon. Carbon monoxide leaves the fuel bed but the oxygen has been largely consumed. Additional oxygen must be supplied as secondary air, to burn the carbon monoxide.

In the combustion of powdered coal there are also the three stages of distillation and combustion, but these phases follow one another in more rapid succession, because of the finer division of the solid coal. The particles of coal vary in size from 0.01 to 0.001 in. Each particle consists of many molecules. The size of an oxygen molecule is about 0.000001 in. It is evident that many oxygen molecules are necessary to surround a coal particle in order to burn it completely. Only a small portion of the coal can be set free as gas in the distillation. The remainder must be burned by direct action of oxygen, as on grates. In order to accomplish this it is necessary to have a large combustion chamber. It is obvious from the above that fine grinding is essential, but that the smallest particle of coal that can be ground economically is still much larger than a molecule of a gas. In this respect gas is the one ideal fuel. Coking coals become plastic on heating and actually may agglomerate in the furnace. Particles of ash often are larger than the original particles of coal.

Pulverized coal is supplied to the furnace with about 30 per cent. of the total air necessary for combustion, as a means for carrying the particles of coal. The secondary air is supplied either in special ports or around the nozzle in the burner. The particles are first heated to their distillation temperature by radiation from the flame. The volatile matter bursts into flame at about 900° F. This flame surrounds the particles and heats them rapidly, driving off all volatile matter, which in turn burns as fast as it meets with oxygen. Because contact with oxygen is essential, it is important that the gases be thoroughly agitated to prevent soot formation and consequent loss of efficiency. Large combustion spaces are necessary for maximum efficiency. About 1 to 2 lb. of coal per cubic foot of combustion space per hour is usual with pulverized coal, while with stokers 1 cu. ft. for 3 to 6 lb. per hour is sufficient.

The ash is either dropped or passes out of the flue. With coking coal the combustible in the flue dust sometimes amounts to 25 per cent. With free-burning bituminous coal it amounts to from 5 to 10 per cent. and with subbituminous coals and lignites the combustible may not exceed 1 per cent. of the dust.

Liquid fuels

Liquid fuels can be burned in two different ways. The first process goes through several intermediate oxidation steps to finally give carbon dioxide and water. The other is a cracking reaction in which the hydro-

carbon breaks down to carbon and hydrogen. The first reaction is the desirable one from the standpoint of maximum efficiency.

Fuel oil must be vaporized or atomized before it is burned. Thus problems of maintaining uniform flow, atomizing the oil and mixing with the proper amount of air are encountered. Various pumps, screens, heaters and regulating devices are necessary.

About two pounds of oil per hour per cubic foot of combustion space can be fired efficiently. After volatization fuel oil burns in the gaseous phase similar to gaseous fuels.

Gaseous Fuels

Gaseous fuels differ from other fuels in that they are burned in their original phase. Since oxygen is a gas and essential to all combustion, it follows that gaseous fuels are the only ones that can be burned in their original state. A hydrocarbon gas, such as natural gas, on combustion unites with oxygen first to form hydroxyl type of compounds which finally break down to carbon dioxide and water.

The size of the particles of gas is extremely small as compared with the possible division of solid or liquid fuels, so the mixture of air with the fuel in combustion can take place readily; therefore high rates of combustion per cubic foot of combustion space per hour are possible. No control of this division is necessary, as in the pulverization of coal or the atomization of oil, and so the mechanics of gas-fired operations are much simplified from this standpoint. In powdered coal installations, other things being equal, the finer the particles can be ground, the better the efficiency obtained. There is a limit of the fineness to which coal can be economically ground. Gaseous fuels are divided into particles of almost infinitely small size.

Comparison of Fuels

In the combustion of coals the impurities must be heated to the temperature of the furnace and this heat is lost when they leave the furnace. One of the impurities found in varying quantities in coal and oil is sulfur. While sulfur burns to sulfur dioxide with the liberation of 4000 B.t.u. per pound, this heat is obtained at the expense of the formation of corrosive products of combustion. Sulfuric acid is the ultimate product of the oxidation of sulfur, in the presence of moisture. Coals vary from 1 to 5 per cent. sulfur. A coal with 1 per cent. sulfur would contain 20 lb. of sulfur per ton, which is converted into 61 lb. of sulfuric acid ultimately, either in the flue gases or in the ashes. A plant using 100 tons of coal per day averaging 3 per cent. sulfur will generate 18,000 lb. of sulfuric acid each day. The small amount of heat furnished by the sulfur is expensive when the resulting corrosion is considered. The ash and moisture content of coals must be heated with heat generated by the combustion of coal. This heat is not available for useful work.

These impurities, excepting ash, are also present in fuel oil and the same losses in efficiency to a greater or lesser degree are found in burning this fuel.

Natural gas contains no appreciable amounts of ash, sulfur or moisture. However, its hydrogen content is high, which lowers the effective heat content, by reason of the heat lost in evaporating the water formed in the oxidation of the hydrogen, as more fully explained under the next heading.

The use of natural gas fuel offers a flexibility of control of the atmosphere of the furnace from an oxidizing to a reducing one. This is especially important in kiln firing, in the pottery, brick and tile industries. The color and quality of the products burned in the kiln are largely dependent on the atmosphere of the kiln. In coal-fired kilns a furnace atmosphere will vary from oxidizing to reducing in a relatively short time. When fresh fuel is added the atmosphere is strongly reducing. As the volatile matter is driven off the atmosphere gradually becomes oxidizing until finally a strong oxidizing atmosphere exists. In stoker-fired furnaces the difference in depth of fuel bed causes changes of atmosphere. In powdered coal and fuel-oil furnaces the atmosphere can be regulated more readily, while gaseous fuel makes it possible accurately and flexibly to control this condition most easily. This is due to the fact that gaseous fuels are in phase with the supporter of combustion and maximum surface contact is possible.

In metallurgical operations a reducing atmosphere is essential to prevent scaling. The scaling loss with some fuels has been found to be greater than the entire fuel costs.

With gaseous fuels only is it possible to preheat the fuel before firing. This added sensible heat increases the efficiency at high temperatures, as preheated air does. Theoretical flame temperatures are lower for natural gas than for coal or oil, but this characteristic is compensated for in practice by preheating the gas and air.

It is possible to burn gas by surface combustion. This is done in the Bonecourt system by feeding an inflammable mixture of air and gas to an incandescent surface. Mixtures can be used with approximately the theoretical quantity of air necessary for perfect combustion. The incandescent surface acts as a catalyst in the combustion of air and gas, and enormous rates of heat production are possible. As high as 700,000 B.t.u. per cubic foot of combustion space per hour have been obtained. This is about 10 times the rate at which coal can be fired.

Measurement of Heating Value

Fuels are compared usually on the basis of heat content per unit as determined experimentally in the calorimeter. Solid fuels are designated as B.t.u. per pound; liquid, as B.t.u. per pound or gallon, and gases on

the basis of B.t.u. per cubic foot. In the experimental determination of the calorific value of a fuel the hydrogen is burned to water, which is condensed by cooling to room temperature. In the burning of the same fuel in a furnace the water formed is not condensed but leaves the stack at the usual stack temperature as a gas or vapor. Thus the latent heat of vaporization of the water formed is lost to the heat absorber. Therefore the experimentally determined B.t.u. value of a fuel is not the actual heat content possible of utilization commercially.

This has given rise to a dual classification of heat values. The experimentally determined value with the water condensed to a liquid is called the gross heat content, while this minus the latent heat of vaporization of the water formed from the hydrogen content of the fuel is called the net value. The two values have been used more with gaseous fuels because of their high hydrogen content. However, all fuels contain some hydrogen, although that remaining in the processed solid fuels such as coke and charcoal is too small to be of significance.

To arrive at a representative comparison of fuels on a direct B.t.u. basis it is necessary to compare them on their net B.t.u. content. It is obviously unfair to compare the gross value of one fuel with the net value of another fuel.

In addition to the heat lost in vaporization of the water formed in combustion, coals have other losses. The B.t.u. content of a coal as quoted is the gross value on a moisture-free basis. This should be calculated to the "as-fired" basis. The loss of sensible heat in the ashes should also be subtracted from the gross value in comparing coal with an ashless fuel. The loss amounts to about 0.1 per cent. of the total heat value for every 2.5 per cent. ash. The firing of coal either in pulverized form or on grates results in the loss of some combustible in the ash either in the stack gases or in that removed from the bottom of the furnace. This sometimes amounts to as high as 25 per cent. of the total combustible in the original coal. The heat content of the sulfur is obtained at the expense of corrosion; in reality it should not be considered of positive value but instead should be considered a detriment in determining the net value of coal.

All fuels must be fired with excess air over the theoretical amount necessary for combustion. Gaseous fuels in general require less excess air than any other fuel. This excess air must be heated to stack temperatures and the heat necessary to do this is not available and thus should be subtracted from the gross value of the fuel. In general boiler-plant practice coals require from 20 to 50 per cent. excess air, oils 12 to 25 per cent. and gases about 10 to 20 per cent.

The moisture content of all fuels must be vaporized and heated to the stack temperatures, and this heat obviously is not available and should not be included in the net effective heating value.

TABLE 1.—*Average Ultimate Analyses of Representative Bituminous Coals*[a]

Condition	Percentages by Weight[b]							B.t.u. per Lb.	Delivered Coal[b]	
	Moisture	Ash	Sulfur	Hydrogen	Carbon	Nitrogen	Oxygen		B.t.u.	No Samples
ILLINOIS										
As received..........	10.85	9.25	2.29	4.28	64.02	1.31	8.00	11,433	11,576	
Moisture free........		10.38	2.57	4.80	71.81	1.47	8.97	12,825	12,554	1,207
INDIANA										
As received..........	11.82	8.87	1.99	4.47	63.91	1.38	7.56	11,384	11,572	
Moisture free........		10.06	2.26	5.07	72.48	1.56	8.57	12,910	12,846	1
EASTERN KENTUCKY										
As received..........	3.46	4.69	1.09	5.18	76.90	1.73	6.95	13,770	13,431	
Moisture free........		4.86	1.13	5.37	79.65	1.79	7.20	14,263	13,848	106
WESTERN KENTUCKY										
As received..........	7.70	9.08	3.24	4.54	67.05	1.43	6.96	12,059	12,191	
Moisture free........		9.83	3.52	4.92	72.64	1.55	7.54	13,065	12,927	152
PENNSYLVANIA										
As received..........	2.80	8.41	2.03	4.86	75.22	1.43	5.25	13,493	14,009	
Moisture free........		8.65	2.09	5.00	77.39	1.47	5.40	13,882	14,260	2,057
WEST VIRGINIA										
As received..........	2.87	6.18	1.25	4.62	79.68	1.41	3.99	14,000	14,164	
Moisture free........		6.36	1.29	4.76	82.03	1.45	4.11	14,408	14,563	1,919

[a] These analyses are averages only. Better coals are available from each of the states named above. Certain selected coals may run up to 12 per cent. more B.t.u. than is indicated by the average figures in the table.

[b] Figures under Percentages by Weight are used in following computations. Delivered Coal figures are shown as check data.

Table 1 shows representative analyses for the bituminous coals of several states. These analyses are weighted average analyses, obtained by taking a number of composite analyses from the latest federal and state publications, weighting each for the number of samples represented, and thus obtaining a grand average for the entire state. Care was taken to select analyses that are thoroughly representative and which give, as accurately as possible, a true average of the commercial coal for each state. In selecting the analyses to be used in obtaining an average, only face and rib samples were considered. No car samples and no slack, run of mine, or outcrop samples are included, and no analyses prior to those appearing for the first time in *Bulletin* 193 of the U. S. Bureau of Mines, published in 1922, have been used. As a further check, there is shown in the table the average B.t.u. content, properly weighted for the number of samples, of all the analyses of coal delivered to the Government from 1915 to 1922, as recorded in *Bulletin* 230 of the U. S. Bureau of Mines, dated 1922. The references for each state are as as follows:

Illinois.—U. S. Bureau of Mines *Bulletin* 193 (1922), and State Geological Survey *Bulletin* 56 (1929). Data from *Bulletin* 193 includes 16 composites, representing 61 samples, and from *Bulletin* 56, data includes samples from 74 mines, covering every coal-producing county in Illinois. The results from each bulletin are averaged to obtain the final average.

Indiana.—U. S. Bureau of Mines *Technical Paper* 417 (1927). Includes 32 composites representing 144 samples.

Eastern and Western Kentucky.—U. S. Bureau of Mines *Technical Paper* 308 (1922). Eastern Kentucky includes 61 composites, representing 251 samples; Western Kentucky includes 30 composites, representing 148 samples.

Pennsylvania.—State Geological Survey *Bulletin* M6, Pt. IV, revised (1928). Includes 22 composites representing 83 samples. Representative analyses from the Pittsburgh, Upper and Lower Freeport, Upper, Lower and Middle Kittanning, Brookville, Clarion, Lower Mercer, Sewickley, Redstone and Blossburg coal beds were selected.

West Virginia.—U. S. Bureau of Mines *Technical Paper* 405 (1928). Includes 13 composites representing 83 samples, checked against 32 composites, representing 110 samples.

Delivered Coal.—U. S. Bureau of Mines *Bulletin* 230 (1923).

Taking the average analysis of the representative Illinois coals as an example, the following calculations show the air required, the products of combustion and heat losses occurring in the reaction of combustion:

Percentage by Weight	As Received, Per Cent.	Moisture Free, Per Cent.
Moisture	10.85	
Ash	9.25	10.38
Sulfur (S)	2.29	2.57
Hydrogen (H_2) (exc. of in moisture)	4.28	4.80
Carbon (C)	64.02	71.81
Nitrogen (N_2)	1.31	1.47
Oxygen (O_2) (exc. of in moisture)	8.00	8.97
Total	100.00	100.00
B.t.u. per pound	11,440	12,825

The air required for burning 100 lb. of this coal will be:

	Amount of Material, Lb.		O_2 per Pound of Material, Lb.		O_2, Lb.	Air, Lb.
Oxygen Required						
S	2.29	×	1.00	=	2.29	
H_2	4.28	×	7.94	=	33.98	
C	64.02	×	2.67	=	170.93	
Total					207.20	
Less O_2 in coal					8.00	
Total O_2 required net					199.20	
Pounds air per pound of O_2					4.33	
Air Required						
Theoretical						862.54
Excess (assumed at 40 per cent.)						345.00
Total per 100 lb. coal (as received)						1,207.54

The water and flue gases formed per 100 lb. of coal may be determined as follows:

	Pounds	Pounds
Water Formed		
Moisture in coal	10.85	
From burning other hydrogen 4.28 × 8.94	38.26	
Total per 100 lb. coal (as received)		49.11

Flue Gases Formed	Pounds		Pounds			
S	2.29 S	+	2.29 O_2	=	4.58 SO_2	
H_2 (exc. in moisture)	4.28 H_2	+	33.98 O_2	=	38.26 H_2O	
C	64.02 C	+	170.93 O_2	=	234.95 CO_2	
N_2 in coal					1.31 N_2	
N_2 in air required	0.769	×	862.54	=	663.29 N_2	
Excess air					345.00 Air	
Moisture in coal					10.85 H_2O	
Total per 100 lb. coal (as received)						1,298.24

The heat losses, having the effect of diminishing the gross heat content of the fuel in the reactions of combustion of 100 lb. of the coal described, assuming 500° F. exit temperature from the stack, are as follows:

					B.t.u. per 100 Lb. as Recd.
1. Latent heat of evaporation:					
Moisture 49.11 lb. of H_2O		970.4			47,656
2. Sensible heat of flue gases:					
Gas	Weight, Lb.	Temperature Difference	Specific Heat		
H_2O liquid	49.11	152	1.00	7,465	
H_2O vapor	49.11	288	0.4686	6,628	
SO_2	4.58	440	0.1494	301	
CO_2	234.95	440	0.2181	22,547	
N_2	664.60	440	0.2464	72,053	
Air (exc.)	345	440	0.2392	36,310	145,304
3. Sensible heat of ash:					
0.1 per cent. of B.t.u. of coal per 2.5 per cent. ash					
0.37 per cent. × 100 lb. × 11,440 as received					4,233
4. Potential heat of combustible lost in ash and flue dust:					
2 per cent. × 100 lb. × 11,400					22,880
Total heat losses in combustion per 100 lb. coal					220,073
Loss per pound					2,200
Net heat available per pound of coal					9,240
Gross B.t.u. per pound (as received)					11,440
Loss, per cent.					19.23

These losses are the actual losses occurring in the reactions of combustion of the fuel—they do not include the furnace losses due to radiation, which are common to all fuels and depend upon the furnace and boiler setting, and not directly upon the fuel. That is, the "net heat available," determined as described above, consists of the gross heating value

TABLE 2.—*Gross Heat Value, Heat Losses and Net Heating Value Available in Various Bituminous Coals*

Per 100 Lb. at 60° F.

	Illinois Coal	Indiana Coal	West Ky. Coal	East Ky. Coal	Pennsylvania Coal	West Va. Coal
Analyses as received						
B.t.u. per lb. (moisture free)	12,825	12,910	13,065	14,260	13,882	14,408
B.t.u. per lb. (as received)	11,440	11,384	12,059	13,770	13,493	14,000
Moisture, per cent	10.85	11.82	7.70	3.46	2.80	2.87
Ash, per cent	9.25	8.87	9.08	4.69	8.41	6.18
Sulfur, per cent	2.29	1.99	3.24	1.09	2.03	1.25
Hydrogen, per cent	4.28	4.47	4.54	5.18	4.86	4.62
Carbon, per cent	64.02	63.91	67.05	76.90	75.22	79.68
Nitrogen, per cent	1.31	1.38	1.43	1.73	1.43	1.41
Oxygen, per cent	8.00	7.56	6.96	6.95	5.25	3.99
Theoretical air required, lb.	862.54	868.42	915.14	1,041.75	1,022.70	1,068.12
Plus 40 per cent. excess air	345.00	347.37	366.06	416.70	409.08	427.25
Total	1,207.54	1,215.79	1,281.20	1,458.45	1,431.78	1,495.37
Flue gases formed, lb.						
SO_2	4.58	3.98	6.48	2.18	4.06	2.50
CO_2	234.95	234.55	246.07	282.22	276.06	292.42
N_2	664.60	669.19	705.17	802.84	787.89	822.79
H_2O	49.11	51.78	48.29	49.77	46.25	44.17
Air (excess)	345.00	347.37	366.06	416.70	409.08	427.25
Total	1,298.24	1,306.87	1,372.07	1,553.71	1,523.34	1,589.13
Heat losses, in B.t.u.						
Water at 60° F. to steam at 212° F.	55,121	58,118	54,201	55,862	51,911	49,576
Flue gases (sensible heat losses at 500° F. stack temperature)						
H_2O	6,628	6,988	6,517	6,717	6,242	5,961
SO_2	301	262	426	143	267	164
CO_2	22,547	22,508	23,614	27,083	26,492	28,062
N_2	72,053	72,551	76,452	87,041	85,420	89,204
Air (excess)	36,310	36,560	38,527	43,857	43,055	44,967
Sensible heat of ash	4,233	4,041	4,341	2,589	4,533	3,500
Combustible in ash and flue dust	22,880	22,768	24,118	27,540	26,986	28,000
Total losses	220,073	223,796	228,196	250,832	244,906	249,434
Loss per pound of coal (B.t.u.)	2,200	2,238	2,282	2,508	2,449	2,494
Percentage loss	19.23	19.66	18.92	18.21	18.15	17.81
Net B.t.u. available	9,240	9,146	9,777	11,262	11,044	11,506

less the above losses. In determining the efficiency of furnace and boiler operation, the radiation losses must be added to the losses stated above before arriving at the heat effective for the ultimate purpose of the furnace, such as producing steam.

The same calculations are summarized in Table 2 for other typical coals.

Similar calculations for several typical natural gases are shown. Table 3 gives the analyses of these gases. The "net heating value" or "net B.t.u." is the gross value minus the latent heat of vaporization of the water formed in the combustion, but without consideration of the sensible heat losses of the flue gases.

TABLE 3.—*Analyses of Typical Natural Gases*

Percentage by volume at 60° F. and 30 in. Hg

	Amarillo, Tex.	Monroe, La.	Ashland, Ky.	Hugoton, Kans.	Columbus, Ohio
Methane	72.94	94.7	75.0	68.86	80.4
Ethane	18.96	2.8	24.0	17.51	18.1
Carbon dioxide	0.39	0.2	0.0	0.10	0.0
Nitrogen	7.71	2.3	1.0	13.33	1.5
Total	100.0	100.0	100.0	100.0	100.0
Specific gravity (calculated)	0.68	0.58	0.68	0.70	0.650
Gross B.t.u. per cubic foot	1,086	1,019	1,197	1,018	1,147
Net B.t.u. per cubic foot	978	913	1,079	917	1,031

Taking the Monroe analysis for an example, we have the following analysis by volume and weight, and the weight of each constituent in one cubic foot of gas.

	Percentage by Volume	Pounds per Cu. Ft.	Pounds per Cu. Ft. Gas	Percentage by Weight
Methane	94.7	0.04244	0.0401907	90.55
Ethane	2.8	0.08033	0.0022492	5.07
Carbon dioxide	0.2	0.11707	0.0002341	0.53
Nitrogen	2.3	0.07443	0.0017119	3.86
Total	100.0	0.31427	0.0443859	100.01

One thousand cubic feet of this gas will require the following theoretical amounts of air for combustion to give the weights of gases listed:

	Weight, Lb.	Air Required, Lb.	CO_2, Lb.	H_2O, Lb.	N_2, Lb.
Methane	40.1907	694.25415	110.3235	90.3487	533.8129
Ethane	2.2492	36.28185	6.5879	4.0463	27.8968
Carbon dioxide	0.2341		0.2341		
Nitrogen	1.7119				1.7119
Totals	44.3859	730.53600	117.1455	94.3950	563.4216

Total weight of air and gas 774.9219 lb. theoretical
Total weight of combustion products 774.9621 lb. theoretical

POUNDS
730.5360 air theoretically required
15 excess, per cent.

109.5804 excess
840.1164 total air required
884.5425 total combustion products with 15 per cent. excess air

One thousand cubic feet of this gas at 1019 B.t.u. per cubic foot has a gross heat content of 1,019,000 B.t.u. The losses in the flue gases and in the latent heat of vaporization are as follows:

SENSIBLE HEAT LOSSES IN FLUE GASES

	Weight, Lb.		Mean Specific Heat		Temperature Difference		B.t.u. Loss
Excess air	109.5804	×	0.2392	×	440	=	11,533.1
Carbon dioxide	117.1455	×	0.2181	×	440	=	11,241.7
Water vapor	94.3950	×	0.4686	×	288	=	12,739.2
Nitrogen	563.4216	×	0.2464	×	440	=	61,083.9

SENSIBLE AND LATENT HEAT OF VAPORIZATION OF WATER

60° F. to Steam, 212° F.

1,122.4 B.t.u. per lb. × 94.3950 lb. =	105,948.9 B.t.u.
Total loss per 1000 cu. ft. of gas	202,546.8 B.t.u.
Net available heat value 1000 cu. ft. of gas	816,453.2 B.t.u.
Net available heat value per cubic foot	816 B.t.u.
Gross heat value per cubic foot	1,019 B.t.u.
Loss, per cent.	19.92

The same calculations for several typical natural gases are summarized in Table 4.

Table 5 shows the analyses of several typical fuel oils from different fields.

Taking the Texas oil as an example, there is in one barrel of oil (42 gal., 316.18 lb.):

	Per Cent.	Lb. per Bbl.
Carbon	85.0	268.82
Hydrogen	12.3	38.90
Oxygen + nitrogen	0.7	2.21
Sulfur	1.7	5.38
B.t.u. per lb		19,230

For one barrel of the above oil we obtain the following products of combustion: (Turn to page 401.)

TABLE 4.—*Gross Heat Value, Heat Losses and Net Heating Value Available in Natural Gas*

Per 1000 Cu. Ft. at 60° F. and 30 in. Hg

	Monroe, La.	Amarillo, Tex.	Ashland, Ky.	Hugoton, Kans.	Columbus, Ohio
Analyses, percentage by volume					
CH_4	94.7	72.94	75.0	68.86	80.4
C_2H_6	2.8	18.96	24.0	17.51	18.1
CO_2	0.2	0.39	0.0	0.10	0.0
N_2	2.3	7.71	1.0	13.53	1.5
Specific gravity (calculated)	0.61	0.68	0.68	0.70	0.65
Gross heating value, B.t.u	1,019	1,086	1,197	1,018	1,147
Net heating value, B.t.u. (gross less latent heat of H_2O)	913	978	1,079	917	1,032
Theoretical air required, lb.	730.54	780.41	860.82	731.71	823.96
Plus 15 per cent. excess	109.58	117.06	129.12	109.76	123.59
Total	840.12	897.47	989.94	841.47	947.55
Flue gases formed, lb.					
CO_2	117.15	130.04	143.84	121.54	136.25
N_2	563.42	605.80	662.63	572.68	634.66
H_2O	94.40	96.99	106.24	91.00	102.86
Air (excess)	109.58	117.06	129.12	109.76	123.59
Total	884.55	949.89	1,041.83	894.98	997.36
Losses, B.t.u.					
Water at 60° F. to steam at 212° F.	105,949	108,860	119,240	102,138	126,044
Flue gases (sensible heat losses at 500° F. stack temperature)					
Water (as vapor)	12,739	13,089	14,337	12,281	15,156
CO_2	11,242	12,479	13,804	11,663	12,818
N_2	61,084	65,678	71,839	62,088	68,807
Air (excess)	11,533	12,321	13,590	11,552	13,008
Total losses	202,547	212,427	232,810	199,722	235,833
Loss per cubic foot	203	212	233	200	236
Percentage loss	19.92	19.52	19.47	19.65	20.58
Net B.t.u. available	816	874	964	818	911

TABLE 5.—*Analyses of Various Fuel Oils*

Based on Dry Oil

Source	Specific Gravity	Baumé Gravity	C	H_2	$O_2 + N_2$	S	B.t.u. per Lb.	B.t.u. per Gal.
Mexico	0.986	12.0	83.7	10.2	4.2	4.1	18,720	153,691
Mid-Continent	0.892	26.9	85.6	12.0	1.1	0.3	19,376	143,950
California	0.971	14.2	84.0	12.7	2.5	0.7	18,820	152,065
Texas	0.907	24.3	85.0	12.3	0.7	1.7	19,230	145,186

WATER

	Pounds
(Assuming no free moisture)	
From burning of hydrogen 38.9 lb. × 9 =	350.1

OXYGEN REQUIRED

Pounds		Pounds		Pounds
5.38 S	× 1	O_2	=	5.38
38.90 H_2	× 8	O_2	=	311.20
268.82 C	× 2.67	O_2	=	717.75
Total O_2 required				1,034.33

AIR REQUIRED

$$\frac{1034.33 \; O_2 \text{ required}}{0.231 \quad O_2 \text{ in air}} = 4{,}477.62 \text{ lb.}$$

Less 2.21 lb. O_2N_2 in oil, 4,475.41 lb. air theoretically required

4,475.41 lb. air theoretically required
1,118.85 lb. 25 per cent. excess

5,594.26 lb. total air required per barrel of oil

FLUE GASES FORMED

Pounds		Pounds		Pounds
5.38 S	+	5.38 O_2	=	10.76 SO_2
38.90 H_2	+	311.20 O_2	=	350.10 H_2O
268.82 C	+	717.75 O_2	=	986.57 CO_2
0.769 N_2 × 4475.41 air			=	3,441.59 N_2
$O_2 - N_2$ in oil			=	2.21 $O_2 - N_2$
25 per cent. excess air			=	1,118.85 air
Total combustion products				5,910.08

The following losses result from the heat of vaporization of the water formed and the sensible heat in the water vapor and other gases at an assumed stack temperature of 500° F.

LOSSES

Moisture (60° F. to 212° F. steam)
1,122.4 B.t.u. per pound × 350.1 lb. H_2O = 392,952.24 B.t.u.

SENSIBLE HEAT IN FLUE GASES AT 500° F.

Gas	Mean Specific Heat		Temperature Difference		Weight, Lb.		Loss, B.t.u.
Water vapor	0.4686	×	288	×	350.1	=	47,248.38
SO_2	0.1494	×	440	×	10.76	=	707.32
CO_2	0.2181	×	440	×	986.57	=	94,675.20
$O_2 + N_2$ from oil	0.2392	×	440	×	2.21	=	232.60
Nitrogen from air	0.2464	×	440	×	3,441.59	=	373,123.42
Air (excess)	0.2392	×	440	×	1,118.85	=	117,756.72

Total sensible heat loss in flue gases	633,743.64
Total loss in combustion of 1 bbl. of fuel oil	1,026,695.88
Loss per pound of oil	3,247
Net available heat per pound of oil	15,984
Gross heat value per pound of oil	19,230
Loss, per cent	16.88

The same calculations for several typical fuel oils are summarized in Table 6.

TABLE 6.—*Gross Heat Value, Heat Losses and Net Available Heat Value in Various Fuel Oils*

Per 42-Gal. Barrel

	Texas Fuel Oil	Mid-Continent Fuel Oil	Mexican Fuel Oil	California Fuel Oil
Analyses				
B.t.u. per pound	19,230	19,376	18,720	18 820
Specific gravity	0.907	0.892	0.986	0.971
Baumé gravity	24.3	26.9	12.0	14.2
Carbon, percentage by weight	85.0	85.6	83.7	84.0
Hydrogen, percentage by weight	12.3	12.0	10.2	12.7
$O_2 + N_2$ in oil, percentage by weight	0.7	1.1	4.2	2.5
Sulfur, percentage by weight	1.7	0.3	4.1	0.7
Theoretical air required, lb	4,475.41	4,369.22	4,585.99	4,777.04
Plus 25 per cent. excess air	1,118.85	1,092.31	1,146.50	1,194.26
Total	5,594.26	5,461.53	5,732.49	5,971.30
Flue gases formed, lb.				
SO_2	10.76	1.86	28.18	4.74
H_2O	350.10	335.79	315.54	386.91
CO_2	986.57	976.84	1,055.82	1,043.49
N_2 from air	3,441.59	3,359.93	3,526.63	3,673.54
$O_2 + N_2$ in oil	2.21	3.42	14.44	8.46
25 per cent. excess air	1,118.85	1,092.31	1,146.50	1,194.26
Total	5,910.08	5,770.15	6,087.11	6,311.40
Losses, B.t.u.				
Water at 60° F. to steam at 212° F	392,952	376,891	354,162	434,268
Flue gases (sensible heat losses at 500° F. stack temperature)				
Water vapor	47,248	45,317	42,584	52,216
SO_2	707	122	1,852	312
CO_2	94,675	93,741	101,321	100,137
$O_2 + N_2$ from oil	233	360	1,520	890
Nitrogen from air	373,123	364,270	382,343	398,271
Air (excess)	117,757	114,963	120,667	125,519
Total sensible heat loss in flue gases	633,744	618,774	650,287	677,345
Total loss in combustion of 1 bbl. of oil	1,026,696	995,665	1,004,449	1,111,613
Loss per pound of oil	3,247	3,202	2,922	3,284
Net available heat per pound of oil	15,984	16,174	15,798	15,536
Gross heat value per pound of oil	19,230	19,376	18,720	18,820
Loss, per cent	16.88	16.53	15.61	17.45

A comparison of Illinois coal, Monroe (La.) natural gas and Texas oil is given in Table 7. No allowance for decreased maintenance and operating labor or for other savings with gas is made in this comparison. The three fuels are compared on a strictly available B.t.u. basis. Thus the gross and the net available B.t.u. contents of the various typical fuels are summarized in Table 8.

TABLE 7.—*Comparison of Available B.t.u. in Illinois Coal, Monroe (La.) Natural Gas and Texas Oil*

		B.t.u. per Lb
Gross B.t.u. of coal, moisture-free basis		12,825
Gross B.t.u. of coal, as-received basis		11,440
Net B.t.u. available		9,240
		B.t.u. per Cu. Ft.
Gross B.t.u. of gas		1,019
Net B.t.u. available		816
		B.t.u. per Lb
Gross B.t.u. of oil		19,230
Net B.t.u. available		15,984
		Cu. Ft.
Natural gas equivalent 1 ton of coal, net		22,647
Natural gas equivalent 1 ton of coal, gross		22,453
Natural gas equivalent 1 bbl. of oil, net		6,195
Natural gas equivalent 1 bbl. of oil, gross		5,967
	Coal per Ton	Gas per 1000 Cu. Ft.
Equivalents on net available B.t.u. basis:	$5.00	$0.2208
	4.00	0.1766
	3.00	0.1325
	2.00	0.0883
	Oil per Barrel	
	$1.00	$0.1614
	0.80	0.1291
	0.60	0.0969
	0.40	0.0646

The net available heating values and not the gross values should be used in the direct and true comparisons of fuels.

In addition to the net available heating content of a fuel the various factors such as flexibility of control of temperature and atmosphere, labor, interest on capital costs of necessary equipment and repairs to furnaces should be taken into account in making a final comparison of fuels.

Actual Comparison of Fuels

Domestic house-heating with coal in hand-stoked furnaces has been studied by the Bureau of Mines and in *Bulletin* 276 the results of 500 tests of such hand-firing operation give an average of about 58 per cent. efficiency.

Efficiencies higher than the above are common with house-heating furnaces designed for gas firing. All house-heating furnaces approved by the American Gas Association Testing Laboratory are required to show an

TABLE 8.—*Summary of Available B.t.u. in Typical Fuels*

Coal	Gross B.t.u. per Pound, Moisture-free	Gross B.t.u. per Pound, As Received	Net Available B.t.u. per Pound
Illinois	12,825	11,433	9,240
Indiana	12,910	11,384	9,146
West Kentucky	13,065	12,059	9,777
East Kentucky	14,263	13,770	11,262
Pennsylvania	13,882	13,493	11,044
West Virginia	14,408	13,994	11,506

Natural Gas	Gross B.t.u. per Cubic Foot	Net Available B.t.u. per Cubic Foot
Amarillo, Tex	1,086	874
Monroe, La	1,019	816
Ashland, Ky	1,197	964
Hugoton, Kans	1,018	818
Columbus, Ohio	1,147	911

Fuel Oil	Gross B.t.u. per Pound	Net Available B.t.u. per Pound
Beaumont, Tex	19,230	15,984
Mid-Continent	19,376	16,174
Mexican, Panuco	18,720	15,798
Bakersfield, Calif	18,820	15,536

efficiency when tested of at least 70 per cent. The convenience of gas is evident and very desirable in house-heating installations.

Industrial applications of natural gas as compared with other fuels follow: In an article in *Power* (Dec. 30, 1930), Bruce shows a comparison of the use of coal and gas in the Memphis Power and Light Company's plant at Memphis, Tenn. The B.t.u. per kw-hr. are shown on the average to be 5 per cent. lower for gas than for coal. Repair and maintenance costs have been reduced 75 per cent. and less operating labor is necessary. Instead of lower superheat, because it was thought that furnace temperatures would be lower with gas, actually 30° F. additional superheat was obtained because of the absence of hard baked ash adhering to the tubes.

Data on open-hearth and pit furnaces in large steel mills in the Pittsburgh district show the following results with natural gas as compared to gases manufactured from coal.

The average results over a period of two years for open-hearth furnaces with a capacity of 80 tons of steel per day are:

Fuel	B.t.u. in Coal Fed to Producers, per Gross Ton of Steel	Furnace Repair, Costs per Gross Ton of Steel
Producer gas	8,760,000	$1.00
Natural gas	6,860,000	0.87

Average producer efficiency 85 per cent.
Western Pennsylvania coal of 13,502 B.t.u. per ton average.
Pennsylvania and West Virginia natural gas average 1150 B.t.u. per cubic foot.
Producer gas average 145 B.t.u. per cubic foot leaving producers at approximately 1380° F.

Thus the B.t.u. used per ton of steel with natural gas as a fuel are about 78 per cent. of those required from producer gas. The furnace repair is about 87 per cent. of that necessary when using producer gas.

Comparisons of natural and coke-oven gas in three plants used for heating pit furnaces with a capacity of approximately 2500 tons of ingots per day show the following results:

	B.t.u. per Ton of Steel	B.t.u. per Ton Required by Natural Gas as Compared to Coke Oven Gas, Per Cent.
1. Natural gas	461,000	48
Coke-oven gas	961,000	
2. Natural gas	840,294	95
Coke-oven gas	881,584	
3. Natural gas	851,808	80
Coke-oven gas	1,058,198	

Group 1 probably represents an extreme case, but gives actual plant data from large-scale operations. Thus from the standpoint of units of heat per ton of finished product, natural gas is considerably more efficient.

Summary

The mechanism of combustion of various types of fuels has been described. It has been shown that the experimentally determined calorific value of a fuel is not the correct criterion for judging its value in comparison with other fuels and that the net available heating value, after subtracting heat not available for useful work, is the true measure of its worth. It has also been shown that other factors, such as flexibility of control and costs of maintenance and repair, enter into any study of the economic utilization of natural gas.

Tables 9 and 10 give the equivalent prices of the various coals and fuel oils as compared to Monroe natural gas, based on the net available heating values of the three fuels.

TABLE 9.—*Equivalent Prices of Coals Compared with Monroe Natural Gas, Based on Net Available Heat Values*[a]

Net Available Heat Value of Monroe Natural Gas = 816 B.t.u. per Cu. Ft.

Price per Ton Coal	Price per 1000 Cu. Ft. Natural Gas	Cubic Feet Gas Equivalent to One Ton of Coal
ILLINOIS COAL.—Average net available heat value 9240 B.t.u. per lb.		
$6.00	$0.2649	22,647
5.00	0.2208	
4.00	0.1766	
3.00	0.1325	
2.00	0.0883	
INDIANA COAL.—Average net available heat value 9146 B.t.u. per lb.		
$6.00	$0.2677	22,417
5.00	0.2230	
4.00	0.1784	
3.00	0.1338	
2.00	0.0892	
WEST KENTUCKY COAL.—Average net available heat value 9777 B.t.u. per lb.		
$6.00	$0.2504	23,963
5.00	0.2087	
4.00	0.1669	
3.00	0.1252	
2.00	0.0835	
EAST KENTUCKY COAL.—Average net available heat value 11,262 B.t.u. per lb.		
$8.00	$0.2898	27,603
7.00	0.2536	
6.00	0.2174	
5.00	0.1811	
4.00	0.1449	
3.00	0.1087	
2.00	0.0725	
PENNSYLVANIA COAL.—Average net available heat value 11,044 B.t.u. per lb.		
$8.00	$0.2955	27,069
7.00	0.2586	
6.00	0.2217	
5.00	0.1847	
4.00	0.1478	
3.00	0.1108	
2.00	0.0739	
WEST VIRGINIA COAL.—Average net available heat value 11,506 B.t.u. per lb.		
$8.00	$0.2837	28,201
7.00	0.2482	
6.00	0.2128	
5.00	0.1773	
4.00	0.1418	
3.00	0.1064	
2.00	0.0709	

[a] Heat values are averages. See footnote to Table 1.

TABLE 10.—*Equivalent Prices of Fuel Oils Compared with Monroe Natural Gas Based on Net Available Heat Values*

Price per Barrel Fuel Oil	Price per 1000 Cu. Ft. of Natural Gas	Cubic Feet Equivalent to One Barrel of Fuel Oil
TEXAS FUEL OIL		
$1.25	$0.2018	6195
1.00	0.1614	
0.80	0.1291	
0.60	0.0969	
0.45	0.0726	
MID-CONTINENT FUEL OIL		
1.25	0.2028	6163
1.00	0.1623	
0.80	0.1298	
0.60	0.0974	
0.45	0.0730	
MEXICAN FUEL OIL		
1.00	0.1503	6655
0.80	0.1202	
0.70	0.1052	
0.60	0.0902	
0.45	0.0676	
CALIFORNIA FUEL OIL		
1.25	0.1939	6445
1.00	0.1552	
0.80	0.1241	
0.60	0.0931	
0.45	0.0698	

Finally, actual large-scale commercial tests of natural gas as compared with other fuels have been listed.

A bibliography of recent articles on the utilization of fuels used in this study follows.

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DISCUSSION

A. L. Brown, New York, N. Y. (written discussion).—Mr. Davis and his associates have prepared a paper essentially consisting of theoretical comparisons of the major fuels. It can be used as a reference on the subject only as a guide to engineers investigating the relative merits of one or more fuels for their practical applications.

I point out this limitation because in the development of the figures shown as "net B.t.u. available" for the several coals, while based on correct combustion equations and numerical data, the fact remains that such figures and all comparative tables of relative value as set forth in the summary can only be correct and conclusive provided the original facts regarding coal are correct.

The weakness of this paper lies, therefore, in the assumption that raw natural products, such as the bituminous coals of this country, can be averaged for chemical quality for definite fields of utilization either by a classification by state boundaries or by any one or more of their chemical constituents. The economic use of coal or its value for certain fuel processes knows no state boundaries. The prevalence in one state of coals from many seams each with different seam characteristics, physical properties and wide variations in percentage of chemical elements makes it impossible to use average chemical composition as a basis of comparison with other fuels and expect the result to be conclusive or of particular value.

The variations in the percentages of "volatile matter" as determined by the so-called "proximate analysis" have a direct bearing on the success or failure of certain groups of coals to meet competition of other fuels for different uses. It is therefore an incorrect premise which the authors have made in establishing values for coals, that the coals of a particular state have been "bunched together" and averaged, including the good, not so good, ordinary and useless, as representative of the state and a composite product which could be selected as the coal that would be sold to compete with natural gas or even oil.

Fuel buyers today are for the most part a discriminating group ably assisted by engineering staffs, and if their processes of fuel utilization are operated as efficiently as their buying procedure, no average coal from any locality will be considered.

The average coal when used is satisfactory for many other reasons than its B.t.u. content alone.

The authors have indicated the source of the data on coal of the several states and the procedure followed in arriving at the average analyses used in their calculations. Undoubtedly the division into the groups named seems adequate to them, inasmuch as many samples and representative seams are included, but I should like to show, by asking a few questions, that the analyses are not representative.

More Detailed Analyses Needed

1. If Eastern and Western Kentucky were compiled separately, why also were not the high-ash and low-ash coals of Pennsylvania and West Virginia?
2. What purpose is served by averaging the high-volatile and low-volatile coals of Pennsylvania and West Virginia?
3. Ohio coals are representative and are used extensively. Why should they not be included and weighted to offset the effect of including two states which have nothing but high-moisture coal?
4. Why should the slack coals of the Pocahontas-New River fields of West Virginia, which are low in moisture and have the highest heat value of any coals, be eliminated from the comparison, especially when these coals have such widespread practical use?
5. How can face and rib samples be more representative of commercial coal than run of mine coals from well-known mine operations?
6. Why should the seam samples as used, admittedly of better quality than delivered coal, be lower in heat value in the figures for Pennsylvania and West Virginia?
7. Can it be assumed that the inclusion of the coal analyses in government and state publications, no matter how carefully recompiled for the preparation of this paper, has any definite relation to the coals accepted by fuel users when an average is used on the basis of chemical quality without consideration of the tonnage of each component grade?

The variations in the B.t.u. values for the five natural gases included in the paper are relatively small compared to the wide differences which have been reconciled in the six coals. To attempt to make fair comparisons using such data is illogical, and in setting up equivalent price values the results are misleading.

I make the suggestion that coal men will not accept the values for "net available B.t.u." for coals or the relative price equivalents of coals to gas until actual analyses of a specified list of coals are compared individually and the maximum and minimum equivalents calculated from these known bases of fact.

Sulfur in Coal

The matter of sulfur in coals as an impurity and corrosive agent is not to be denied. Coal producers go to great lengths to keep this down to a minimum amount. This effort has not been wholly due, however, to the demand to reduce it because of its chemical activity but rather to improve the storage and burning performance qualities of the coals.

The sulfur in coals is not wholly destructive or corrosive, inasmuch as a large portion of this element is recombined with the basic ash constituents and is not evolved. The eventual conversion of the sulfur in the ash or clinker by atmospheric action and soil contact to useful products is every-day chemistry. The removal of sulfur gases from the burning or processing of coal is effected at a profit in some plants and in most such plants its corrosive forms are neutralized.

It is entirely too positive a statement, and one not supported by good chemistry, to say that all sulfur in coal will generate its full equivalent in sulfuric acid when the coal is burned.

Comparison of Fuels

In the section on Actual Comparisons of Fuels (p. 403) the examples given undoubtedly can be duplicated in other plants. However, in each case the mere statement of the facts is not conclusive evidence that they can be applied generally to the industry or class of consumer in question. Complete details of the comparative use of the two fuels might readily disclose other contributing reasons for the showing made with natural gas as against producer or coke-oven gas than the single item of the relative heating efficiency of the natural gas itself.

In defense of coal also in areas where both fuels are easily and cheaply obtained, it has been my privilege to witness the operation of two plants making identical products, one with natural gas, the other with producer gas. In one case natural gas is preferred, in the other natural gas has been rejected in favor of producer gas by improvement in plant equipment which has decreased the fuel cost per unit of finished product.

To prevent misrepresentation of facts and to reach a clearer understanding of the relative inherent B.t.u. values of fuels, I concur with the authors in the necessity for making fuel B.t.u. comparisons from the same basis of calculation. Too many times coals are analyzed on an "as-fired" basis with the heating value accompanying it quoted on the "dry" basis.

Sensible Heat Loss

The sensible heat loss in the ashes from coal burning as stated in the paper is debatable. This loss is almost entirely controllable in present-day solid fuel burning.

The authors have pointed out the heat losses with coal burning so completely that it might be only fair to state that in the transmission and distribution of high-pressure natural gas which has been stripped ("dry gas," so-called), it has been found necessary to "fog" water or oil vapor into the gas to prevent troubles from dust, scale, etc. This increase in humidity should be shown as an additional heat loss in burning such gas and will decrease the "net available B.t.u." of natural gases.

The effective heat of coal today has been increased in modern boiler plants by reducing the sensible heat loss through the use of economizers and air preheaters. With the larger amount of excess air used in burning coal, the relative saving through the use of this equipment applies to a larger degree to coal than to natural gas, and therefore increases the net available B.t.u. of coal.

Comparison of Costs

In the summary of the paper, the answer to the coal versus natural gas question is clearly shown. The matter is and always will be largely one of dollars and cents per unit of product, which for all major fuels is closely related to the B.t.u. per unit of fuel. Therefore the latter governs more strongly than other factors.

On page 406, $2 West Virginia coal is listed as equivalent to $0.0709 gas. It is a matter of record in present fuel markets in some parts of West Virginia that coal is $2 (delivered price), while natural gas to perform the same quantity of work is selling near 19¢. At this latter figure gas would be equivalent to approximately $5.50 coal. As the freight charges are less than $1 in the case mentioned, the coal is worth something more than $4.50 per ton at the mines. This figure compared to the present price of $1 unquestionably shows the real worth of coal at present-day fuel prices. Similar situations can be found in other states and distributing territories for both fuels.

The conclusion is obvious; either gas is valued too highly as a competitive fuel or coal is not commanding a fair price. Economic conditions point to the latter statement as the one more nearly correct.

Practical Tests Needed

W. Mittendorf, Cincinnati, Ohio (written discussion).—While the title of this paper is "Economic Utilization of Natural Gas," apparently it presents principally an interesting study of comparative heat values of gas, oil and coal as determined from a laboratory standpoint, together with an explanation of the theoretical losses obtained from these analyses and applied to the burning of these various fuels.

The authors have dwelt at length on the mechanism of combustion of coal and to a lesser degree on gas and oil, all of which are more or less known today, but in the final analysis a true comparison of these fuels can be obtained only by actually burning them under the conditions for which they are to be utilized and thereby obtaining information as to the unit cost of production, the pounds of water evaporated per unit of fuel, the over-all efficiency or any information relative to the comparison, or by burning these fuels in a furnace designed for burning each individual fuel and obtaining the same information.

The paper shows what they term actual losses occurring in the reactions of combustion of the fuel for coal of 19.23 per cent. and of gas 19.92 per cent., which do not include radiation and other unaccounted-for losses. Deducting this 19.23 per cent. on coal from 100 per cent. leaves an efficiency of 80.77 per cent., which according to these tests cannot be made higher because they are fixed losses, when as a matter of fact many accurate evaporative boiler tests have been made with coal showing boiler and furnace efficiencies higher than this and including radiation and unaccounted-for losses but not over-all efficiencies.

In their comparison of fuels, stress is laid upon the sulfur content of coal and resulting corrosion. Sulfur in coal is mostly burnt to SO_2 and in the many years that coal has been used under boilers, and considering the long life of boilers in actual use, corrosion due to sulfur has not been a serious cause for trouble. On the other hand, in the burning of natural gas much water is formed because of its very high hydrogen content, and efficiencies are affected by the expansion of this water into larger volume than would be produced by the regular products of combustion causing an increase in velocity of gases through the boiler reducing the effective heat transfer. Also, one of the largest contributing causes to corrosion in boiler settings is dampness, and the large amount of water produced in the burning of natural gas is not conducive to prevention of corrosion.

As explained in the paper in the section on Mechanism of Combustion of Coal, it is not possible to get enough air through grates to support combustion. It is this condition that offers a flexible control in kiln firing by hand, as the amount of air can be regulated at all times by the over-fire air to produce a reducing atmosphere. At the same time, coal has a distinct advantage in this class of work by being able to produce a long flame under this condition to carry the heat to the distant parts of the kiln.

There are many metallurgical operations in which gas has an advantage and there are many of these operations in which coal has an advantage. For example, the burning of coal is advantageous in malleable iron work, in which all the melting and annealing is done by coal and in which gas has never been successfully used to the present time.

While it is possible to preheat the fuel when burning gas, it is more necessary to do this with gas than with coal, for the reason that the volume of gas is much larger for a given amount of combustion products, although coal is now being preheated in pulverized form before it is admitted to the fire.

While surface combustion has not as yet come into general use, some of the first experiments in England by Bone were made on coal.

For years there has been much discussion on the question of reporting a fuel on the basis of gross or net heat value, and as far back as 1921 the writer has come in

contact with it in South America, but up to the present time there has been no real decision on this question, as both methods have their distinct advantages from different viewpoints.

The net values have been reported, however, with only a reduction from the hydrogen content burning to water and no deductions for any other reasons.

Commercially it would be difficult and expensive for a user of large quantities of coal who obtains his supply from various sources to make an ultimate analysis on the many samples of coal made in checking his supply, and in order to obtain the net value this kind of an analysis would be necessary. Even in the same seam of coal as mined the hydrogen content will vary and a sample taken from certain parts would not be a representative average of the total cost at all times.

Ultimately, again, the user of fuel is interested principally in the results of burning the fuel, and uses an analysis for check purposes, and for this purpose the gross heat value, which can be readily obtained, is a measure of this.

The paper, in quoting the loss due to combustible in the ash, says that this loss sometimes amounts to as much as 25 per cent. of the total combustible in the coal. When these conditions obtained there must be some extenuating circumstance connected with it and the plant should be reconditioned, as with this kind of operation the efficiency would be low no matter what kind of fuel was burned. If 25 per cent. of the original combustible remains in the ash, it would follow that there would be 75 per cent. of unconsumed fuel in the ash, whereas with the present methods of burning coal 10 per cent. would be a fair figure.

Operating under equivalent conditions and with the present-day methods of burning coal, there is no greater amount of excess air necessarily used in burning coal than with gas, as is indicated by an analysis of the flue gas in every-day operation in plants equipped to obtain this information.

While the analyses of the coals as quoted for comparison with gas represented an average of the whole state at the time the samples were made, they would hardly be comparable with the analyses of the same state at the present time. *Bulletin* 193 of the U. S. Bureau of Mines was published in 1922 and the samples were probably made in 1921. *Bulletin* 308 was made in the same year. *Bulletin* 230 was made in 1923.

Specifically, it is the writer's knowledge that new mines have been opened in seams of high-quality coal in the Harlan County field of Eastern Kentucky, and mines of coal of lower quality have been worked out. With these changes the average quality of the coal has improved in this field since the bulletin was published. The same conditions have obtained to a greater or lesser extent in the other fields, as coal generally is of a better quality since these bulletins were published.

However, a user of coal does not purchase coal of average quality in any field but usually obtains the coal best suited to his purpose, and this may be a coal of a higher or lower grade from a certain mine or county, so that an average analysis of a whole state or section would not be comparable with a coal best fitted for his use.

Losses in Burning

Based on the contention that coals are now being burned with the same amount of excess air as gas and by the same reasoning that the excess air would be 25 per cent. in either case, increasing the amount quoted in the paper from 15 to 25 per cent. for gas and decreasing the amount from coal from 40 to 25 per cent., and also based on the contention that the combustible in the ash from the coal will be between 10 and 12 per cent., the following calculation shows by the author's method of figuring the actual losses that the percentage of loss in the burning of the coal would be 17.95 per cent. instead of 19.23 per cent., and that with the losses in the burning of the gas would be 20.7 per cent. instead of 19.92 per cent.:

Latent heat of evaporation

	B.T.U. PER 100 LB. AS RECEIVED
1. Moisture 49.11 lb. of H_2O × 970.4	47,656

2. Sensible heat of flue gases:

Gas	Weight, Lb.		Temperature Difference		Specific Heat		
H_2O (liquid)	49.11	×	152	×	1.00	=	7,465
H_2O (vapor)	49.11	×	288	×	0.4686	=	6,628
SO_2	4.58	×	440	×	0.1494	=	301
CO_2	234.95	×	440	×	0.2181	=	22,547
N_2	664.60	×	440	×	0.2464	=	72,053
25 per cent. excess air	293	×	440	×	0.2382	=	30,837

3. Sensible heat of ash

0.1 per cent. of B.t.u. of coal per 2.5 per cent. ash	
0.37 per cent. × 100 lb. × 11,440 as received	4,233

4. Potential heat of combustible lost in ash and flue dust

1.2 per cent. × 100 lb. × 11,400	13,728
Total heat losses in combustion per 100 lb. of coal	205,448
Loss per pound	2,054
Net heat available per pound coal	9,386
Loss, per cent	17.95

730.5360 theoretical air required
25 per cent. excess

182.6340 excess
913.1700 total air required
957.5559 total combustion products with 25 per cent. excess air.

One thousand cubic feet of this gas at 1019 B.t.u. per cubic foot has a gross heat content of 1,019,000 B.t.u. The losses in the flue gases and in the latent heat of vaporization are as follows:

Sensible heat losses in flue gases:

	Weight, Lb.		Mean Specific Heat		Temperature Difference		B.T.U. Loss
Excess air	182.6340	×	0.2392	×	440	=	19,221.8
Carbon dioxide	117.1455	×	0.2181	×	440	=	11,241.7
Water vapor	94.3950	×	0.4686	×	288	=	12,739.2
Nitrogen	563.4216	×	0.2464	×	488	=	61,083.9

Sensible and latent heat of vaporization of water (60° F. to steam at 212° F.)

	B.T.U.
1,122.4 B.t.u. 1 lb. × 94.3950	10,5948.9
Total loss per 1,000 cu. ft. of gas	21,0235.5
Net available heat value per 1,000 cu. ft. of gas	80,8764.5
Net available heat value per cubic foot	808
Gross heat value per cubic foot	1,019
Loss, per cent	20.7

With the net heat available based on the above calculations, the equivalent price on Illinois coal compared with Monroe natural gas would be as follows:

Net available heat value of Monroe natural gas, 808 B.t.u. per cubic foot
Net available heat value of Illinois coal, 9386 B.t.u. per pound as received

Price per Ton Coal	Price per 1000 Cu. Ft. Natural Gas	Cubic Feet of Gas Equivalent to One Ton of Coal
$6.00	$0.2582	
5.00	0.2152	23,232
4.00	0.1721	
3.00	0.1291	
2.00	0.0860	

Domestic house heating with coal is now undergoing a distinct change with the advent of the household stoker and efficiencies are increasing, tests having been made with the small stoker with the proper coal for its use showing saving in actual heating costs of more than 30 per cent. over oil and gas.

Advantages of Coal over Gas

Coal has a distinct advantage over gas in the matter of reliability of supply. Stocks of coal can be carried to prevent any shutdowns in the event of interruptions to the supply. In the event of interruption to the supply of gas, nothing can be done to keep a plant operating, and it is for this reason that public utility plants when using gas generally equip their boilers with gas on the front or back side and with stokers or pulverized coal on the other side, as the case may require.

The writer knows of a public utility corporation that sells gas commercially, and in its own boiler plant the boilers are equipped for both gas and coal.

That coal can be burned more efficiently than gas or oil under conditions that are comparable was demonstrated recently in tests made to show over-all efficiencies under conditions of accuracy that cannot be questioned.

Burning coal, the Detroit Edison Co. showed an over-all efficiency of 90.5 per cent., while the Pacific Gas & Electric Co. in California showed an over-all efficiency of 87 per cent. on apparatus designed to burn oil and an over-all efficiency of 84 per cent. on apparatus designed to burn gas.

Excess Air Needed

C. V. Beck, St. Louis, Mo. (written discussion).—I object seriously to the main premise of this paper. The whole idea is built around the fact that natural gas can be burned with 15 per cent excess air, oil with 25 per cent excess air and coal with 40 per cent excess air. This basic assumption is not warranted and is put forth in a manner that would assume these to be generally accepted facts beyond contradiction. I have never heard of natural gas being burned with 15 per cent excess air experimentally, much less commercially. One paper I have read—a description of the performance of the new plant of the Pacific Gas & Electric Co., by F. G. Philo—mentions the fact that it is possible to burn natural gas with 15 per cent excess air, but even in this paper he does not state that it is being done either under test conditions or in general practice. Furthermore, if the Pacific Gas & Electric Co. has been able to burn gas with 15 per cent excess air it should be thoroughly noted that this is a superpower plant built especially to burn natural gas, and of the very latest design. On that assumption it is entirely misleading to compare the maximum results obtained by the most modern gas-burning plant with ordinary results with other fuels obtained in average commercial plants.

Harry Perkins, until recently the combustion expert of the Gulf Refining Co. at Port Arthur, one of the largest users of natural gas, only recently informed me that this company was never able to obtain complete combustion even experimenting with 25

per cent excess air, and found it necessary to operate commercially with no less than 40 per cent excess air.

If the very best obtained excess air is to be compared with other fuels it should be in similar modern plants. Powdered coal is being burned with considerably less than 25 per cent excess air at the present time in a number of central stations, and even commercially. Anheuser-Busch, in St. Louis, at the present time is burning the lowest grade of Illinois coal with slightly under 25 per cent excess air as an average. Detroit-Edison, which is a comparable installation to that of the Pacific Gas & Electric Co., is burning powdered coal with much less than 25 per cent excess air and has attained over 90 per cent boiler efficiency on coal—something never attained with natural gas or oil as yet.

The assumption, which seems very plausible, that natural gas due to its gaseous state enables a perfect air mixture very readily, is absolutely erroneous. This is another idea advanced in this paper as an established fact, and passed over as if it were beyond contradiction. The fact is there is nothing more difficult than gas with which to achieve perfect air mixture. This is a fact that is well established by many impartial natural gas experts, the reason for which is somewhat in debate. However, the consensus of opinion seems to be that natural gas and air have a greater tendency to stratify than any other fuel. The inertia of powdered coal or oil allow them to be thrown through air streams, as it were, and thereby makes possible an intimate air mixture that cannot be so easily achieved with natural gas. Of course, for stoker-fired coal this idea would not apply, but it is well to note that with stoker-fired coal combustion is far along when the fuel and air reach a state similar to that of gas firing.

Sulfuric Acid Not a Factor

The author's ideas of sulfuric acid are weird and extreme. It is impossible for sulfuric acid to form at boiler or stack temperatures. Under actual burning conditions there is no corrosive effect from sulfur. Sulfuric acid is not formed until the sulfur dioxide fumes, which are not corrosive, are discharged into the atmosphere and reduced in temperature below 212°, and come into contact with moisture in the air. Therefore sulfuric acid is absolutely out of the picture as far as the plant is concerned. It is not formed in either the furnace or the stack and if it is formed in any appreciable quantities it is long after the fumes have left the stack. Furthermore, SO_2 and water form sulfurous acid—a mild bleaching agent—not sulfuric acid.

Heat Loss and General Analyses

As mentioned in the two preceding discussions, surface combustion from an incandescent surface was first accomplished with coal. It is not generally used in commercial practice evidently owing to the expense of maintaining refractories under such a scheme. In this paper a sensible heat loss is figured for coal for heat in the ash. This heat in general is recovered in the draft air and therefore it is not a loss.

The preceding commentators have taken considerable exception to the coal analyses used. As the main sample in this paper was taken from average Illinois coal, with which I am intimately familiar, I desire to add the following: Face samples, particularly with Illinois coal, are not average samples. Illinois coal contains several bands. In the hand mining process these bands are removed first by the miner and further picked out over picking tables. In mechanical mining the entire amount of these bands is removed by picking tables. In mechanical mining the amounts eliminated vary from 5 to 10 per cent of the total product mined.

While I assume that the author of this paper has averaged fairly the analyses quoted, it is my impression that 4.28 per cent net hydrogen on an as-received basis is too high; I would not take serious exception to this point without further study, but suggest that it is worth while looking into.

It is manifestly unfair to take coal as received on the basis of face samples containing the maximum possible impurities and compare it with gas and oil absolutely pure when, as we all know, both of them contain moisture in various proportions.

Also, a loss is figured off for coal due to combustible in the flue dust, whereas perfect combustion is assumed with both gas and oil. An analysis of any gas-fired furnace will show unburned methane in the fuel gases, generally approximately 3 per cent.

The loss figured off by Mr. Davis and his coauthors for water vapor is figured at 288° from 212°. No loss is figured for heating the oxygen in the air or the hydrogen in the gas from 60° to 212° or, in other words, heating these products 152°. While this is a very small loss, it should be figured, inasmuch as the author has figured off for coal everything he could think of.

In burning oil there are many losses not mentioned in this paper, the largest of which is that oil is commonly atomized with steam, generally requiring one pound of steam per pound of oil fired. Neither the cost of the steam nor the loss of passing this steam through the fire is mentioned.

Domestic furnace gas firing with modern gas appliances is compared against hand firing. If gas goes into a house, a modern gas burner as of a consequence goes along with it. If automatic firing of gas is to be compared with coal, why not compare them on the same basis—automatic firing versus automatic firing? There are 50-odd makes of stokers on the market, so this is no new or revolutionary method of firing coal. It is unfair to compare gas fired in the best way with coal fired in the worst way.

Excess Air Most Important

All of the objections named I regard as minor objections to the one first mentioned; *viz.*, the net assumption that gas is fired with 15 per cent excess air and coal with 40 per cent. The plants that are using 40 per cent excess air on coal will doubtless use 40 per cent or more excess air with gas. Comparing similar installations, it will be found that the excess air with gas and coal does not materially differ, and if anything the break is in favor of coal. It is certainly highly deceptive to compare performance of the best superpower plants especially constructed for gas with the average carelessly operated coal plant containing mediocre and somewhat ancient equipment, and assume each as a general average. The paper gives natural gas preference on every item and does not present the relative facts.

R. D. Hall, New York, N. Y. (written discussion).—Congratulations are due to Mr. Davis and his associates for having tackled the most timely topic among producers of various kinds of fuel. The conclusions which this article draws are so important that one cannot allow them to pass without comment.

In discussing the declarations on pages 390 and 391 regarding the relative readiness of coal and gas to burn under circumstances most greatly favoring such combustion, it may be said that the most recent students of the subject declare that coal when burned most economically requires about 25 per cent of excess air and gas about 15 per cent, that there are advantages in a relatively stationary or slowly moving fuel such as fine coal when scrubbed by a rapid current of air and that gas when intimately mixed with air tends, when burning, to befoul itself, leaving unburned methane in the chimney gases.

Calorific Values of Coals

The authors of the paper have laid much stress on the correctness of the figures they have obtained for the calorific value of coals in the various states, but it will be necessary to delay final judgment till averages based on a more complete inquiry loaded in a scientific manner have been made. Too much weight has been laid on certain

composite figures which happen to be those of the U. S. Bureau of Mines. These composite figures represent merely the averages of a number of analyses from a single location. In consequence they have, in a sense at least, the weight of only a single analysis. Thus 32 composites from Indiana representing 144 samples should be rated as 32 samples only. In Pennsylvania, also, the authors take analyses representative of the various beds, regardless of their importance. The Lower Mercer has long been completely worked out; it never had more than a very few mines. The Redstone is not being operated. The Bloss bed is of insignificant extent and largely worked out. Such unweighted results give an incorrect impression except when taking all analyses. When that is done the needed correction will be applied by the fact that the number of analyses will be an index of the importance of the various seams and of the degree of their operation. What is wrong in making selection may be permissible when all analyses without selection are taken.

I have taken an average of all the as-received calorific values of face samples of Indiana coals, excluding such analyses as were not taken in accord with the regular Bureau of Mines standards, so designated in *Technical Paper* 417. Thus I have taken 218 samples in Indiana as against the authors' 144. Their figure is 11,384 B.t.u. and mine 11,656 B.t.u. This shows what errors may spring from an incomplete study of the published analyses. This is the only complete state study I have thus far made.

It is pleasing to note the authors' footnote to Table 1: "These analyses are averages only. Better coals are available in each of the states named above." To this I might add that the best coal sample in Illinois showed on analysis 11.9 per cent more B.t.u. per pound of bed sample as received than is credited by the authors, Indiana 10.4 per cent more, Eastern Kentucky 7.5 per cent more, Western Kentucky 11.9 per cent more, Pennsylvania 8.7 per cent more, and West Virginia 11.1 per cent more. That these samples while exceptional are not out of line with some of the good coals of the state may be judged from the fact that the average as-received values of face samples of the coals of McDowell County, West Virginia, run 3.9 per cent (575 analyses) more than those given by the authors, the coals of Raleigh County, in the same state, 4.4 per cent more (311 analyses) and the coals of Mercer County, also in the same state, 4.8 per cent more (190 analyses).

It must be remembered that face samples are supposed to exclude only the dirt that the miner could be expected to reject. Apparently the average coal actually being produced has much less ash than the average for the whole resources of the state, (1) because the dirty beds and parts of beds have ceased to be producers or have taken a less conspicuous role, (2) because the dirty parts of those beds are not merely cleaned, they are left undisturbed in their entirety with much salable but under-standard coal with them, (3) because the coal is better cleaned below ground and above ground than was anticipated. Mechanical cleaning has made a great improvement in the quality of marketed coal. Selective mining is making a still further improvement.

It is interesting to note that in nearly every case given in Table 1 the delivered coal has a higher calorific value, as received, than the face sample as-received coal. This is partly due to the loss of moisture in transit, but it is significant nevertheless that even before 1923, when *Bulletin* 230, quoted in the paper, was published the average delivered coal received by the Government was superior to the face sample coal as received. That difference is doubtless greater today.

Some day better averages will be available; until that time one must be chary of criticism of the authors' figures even though they have taken only 45 composites out of 453 in their West Virginia figures and have omitted all the other analyses.

Much depends, in the contest between natural gas and coal, on whether the gas will be burned in equipment suited to that fuel. After all, the number of B.t.u. is

not as important as the use made of them. Methane gives only about 15 per cent of its heat as radiant heat. If the convective heat therefore is not taken care of, natural gas gives little or no heat to the boiler. In even the least efficient equipment some of the convective heat is used, but there is always a probability with a boiler not designed for the using of gas that a large percentage of the convective heat will be lost.

Improvements in Preparation and Use of Coal

C. E. Lesher, Pittsburgh, Pa. (written discussion).—We have made a careful study of this paper and have considered it in connection with other current published material on the relative efficiencies of coal and natural gas as fuel for raising steam. The authors seem to be particularly interested in proving that natural gas has no equal in this field. Their conclusion is based upon the very latest data on the use of gas, based upon tests made under best possible operating conditions, as compared with the use of coal under boilers years ago. There has been notable development in economical firing mechanism for coal in recent years, with the results of which the authors are not familiar. Efficiencies in burning coal have reached 85 per cent continuously, and even higher under conditions that would make it difficult to establish that the use of natural gas would prove to be an economy.

The authors depend for their data on efficiency in the combustion of coal upon published material dated 1915 and 1928, since when there has been much progress not only in the firing and combustion of coal under boilers, but in improvement in the quality of the coal through highly mechanized mining and cleaning at the mines. Furthermore, price relationship between coal and natural gas year by year has favored the use of coal; that is to say, the price of coal has been going down.

This controversy—for controversy it is—between the proponents of coal and natural gas will continue and perhaps never be finally settled, because in every locality, depending upon the combustion equipment, upon the quality of coal available and the freight rate on that coal as against the delivered price of natural gas, there will be a definite relationship that will, for that set of conditions in that locality, determine which is the most economical.

What we need, and what the American Institute of Mining and Metallurgical Engineers should foster and publish, are comprehensive, accurate engineering comparisons that will enable a prospective user to determine for his particular condition which is the most economical, bituminous coal or natural gas.

State-wide Analysis Versus Analysis by Coal Seams

H. N. Eavenson, Pittsburgh, Pa. (written discussion).—The writers of this paper have done a large amount of work in preparing the data which they have submitted with it and are to be congratulated on the excellent way in which this has been done.

Among coal men there is a very evident feeling, as shown by the discussion at various places, that the use of state-wide analyses is of no particular value. The point of view of the authors, of course, is that they had to have some basis for comparison, and it would have been a much larger job to have prepared the figures on the basis of seams rather than on geographical divisions.

For some other purpose the writer found it necessary to prepare an average analysis of the low-volatile coal in the southern West Virginia field, and, as that state contains a large area of low-volatile coal the heating value of which differs considerably from the figures shown in Table 1, where both high-volatile and low-volatile coals are included in the average, Table 11 is submitted.

TABLE 11.—*Average Analysis of All Seams, Low-volatile Field of Southern West Virginia*

Number of seams—9
Number of samples—1733

	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	B.t.u., Per Cent.
Average analysis	2.59	18.07	74.29	4.96	0.66	14,530
Maximum of seam	2.68	21.35	75.21	7.58	1.06	14,742
Minimum of seam	1.61	15.97	72.35	3.65	0.59	14,162

The above analyses include all of those given by both the U. S. Bureau of Mines and the West Virginia Geological Survey having less than 23 per cent volatile matter and are on as-received basis.

Similar data are not available for Pennsylvania, where the same condition exists, but the low-volatile coal in Pennsylvania is comparable in heat value to that in West Virginia, although its ash and sulfur content will average somewhat higher.

AUTHOR'S REPLY TO DISCUSSION

R. E. DAVIS (written discussion).—We appreciate fully that any buyer of large quantities of coal, in choosing between a given natural gas and a given coal, would find it necessary to make a careful check of their relative merits as fuel, and we believe that we have pointed out the proper method of doing this.

It is self-evident that in choosing between a given coal and a given natural gas the merits of the *particular* coal and of the *particular* gas must be measured, and nowhere in our presentation of the subject do we suggest any other procedure. It is also apparent that the conditions under which the coal or the gas will be used should control, and that no consideration need be given to theoretical efficiencies that might be obtained from the use of either fuel under conditions that do not exist. Any individual user of fuel having a plant in operation will consider what he can do with any one of several fuels, and he certainly will not be governed by any averages or by any extremes.

In presenting a comparison of certain fuels we have attempted to illustrate the method of comparison by using what we believe are average coals compared with an average natural gas, and assuming that each is burned under average conditions as they exist today. Certainly better coals are available than the average used in our calculations. So also are better natural gases available. More efficient burning apparatus can be built, and where new plants are being considered higher efficiencies will be sought than are assumed in our calculations for the average present-day plants.

It is comparatively easy to measure the relative direct heating values of various fuels. It is not so easy to measure the actual value accruing from those qualities of natural gas which are reflected in the convenience of its use, its cleanliness and the flexibility of its control. These latter factors have much greater value in some situations than in others, and in any given situation should be given the weight that the particular circumstances justify.

Much of the criticism of our paper, we believe, is based upon a misinterpretation of t, as will be revealed by a careful reading of both the paper and the criticisms.

INDEX

(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

I